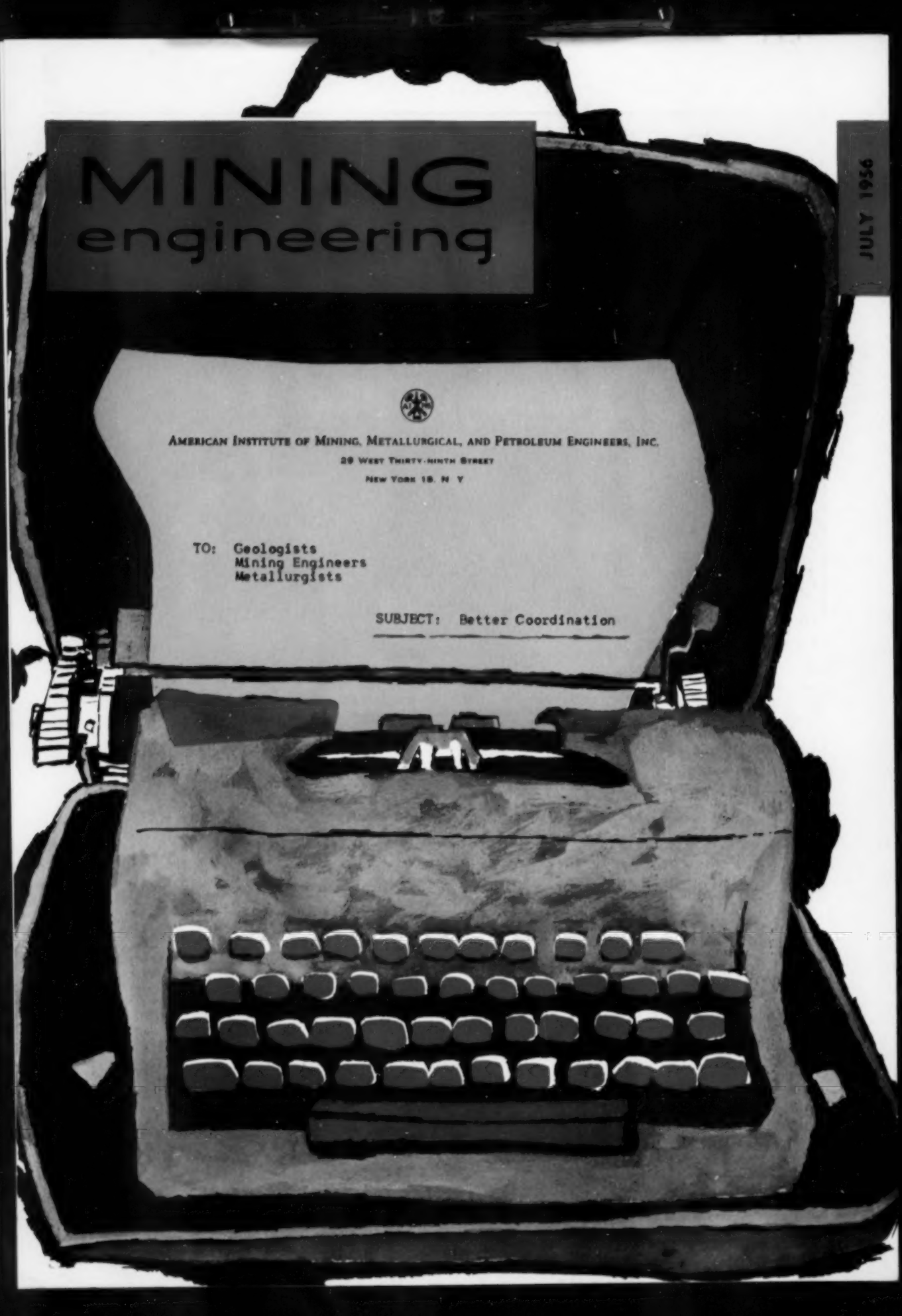
MINING engineering

JULY 1956



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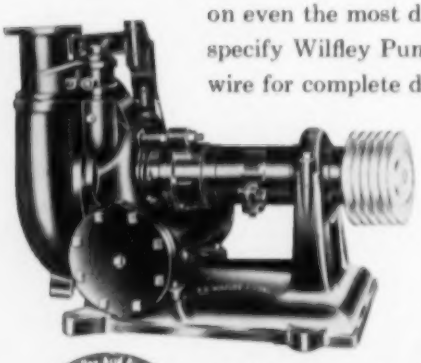

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MINING engineering

VOL. 8 NO. 7 JULY 1956

COVER

Artist Herb McClure chose a typewriter as the symbol of the interchange of information and ideas between specialists that is so necessary in today's complex undertakings. The portable—perched on someone's knees who is on a train or a plane, or is out in the field—seemed to symbolize this flow of ideas between geologists, mining engineers, and metallurgists. Turn to page 698 for a deeper discussion of these ideas.

FEATURES

Personnel	662	Drift	697
Books	668	AIME News	745
Letters	670	1957 AIME Nominations	746
Manufacturers News	675	Personals	751
Reporter	679	Professional Services	758
Mining News	685	Coming Events	760
Trends	690	Advertisers Index	760

ARTICLES

SYMPOSIUM: Greater Cooperation for Metallurgical Planning 698

Metallurgist's View on Milling Base Metal Ores — F. W. McQuiston, Jr. 699

Metallurgical Planning for Porphyry Copper Ores — J. B. Clemmer 701

Geologist-Metallurgist Cooperation in Porphyry Copper Exploration — Kenyon Richard 703

Metallurgical Planning in Connection with Lead-Zinc or Lead-Zinc-Copper Mines — M. K. Kildale 706

Mining Engineer's View of Cooperation — A. G. Kirkland 707

Summary of Symposium — Reno Sales 709

Economic Factors in Cold Weather Operations — E. B. Spice 712

Mining Engineering Notebook
Use of Belts and Concrete Ore Passes — E. G. Williams 718

TRANSACTIONS

Cananea's Program for Leaching in Place — R. C. Weed 721

Raw Materials Preparation at the Brandon Plant, Mississippi — J. C. Holm 724

Tailing Disposal at the Morenci Concentrator — P. F. Allen 729

Mining Methods at the Iron King Mine — H. F. Mills and L. Bombardieri 732

Reagent Control in Flotation — C. H. Bushell and M. Malnarich 734

A Kinetic Study of the Leaching of Molybdenite — W. H. Dresher, M. E. Wadsworth, and W. M. Fassell, Jr. 738

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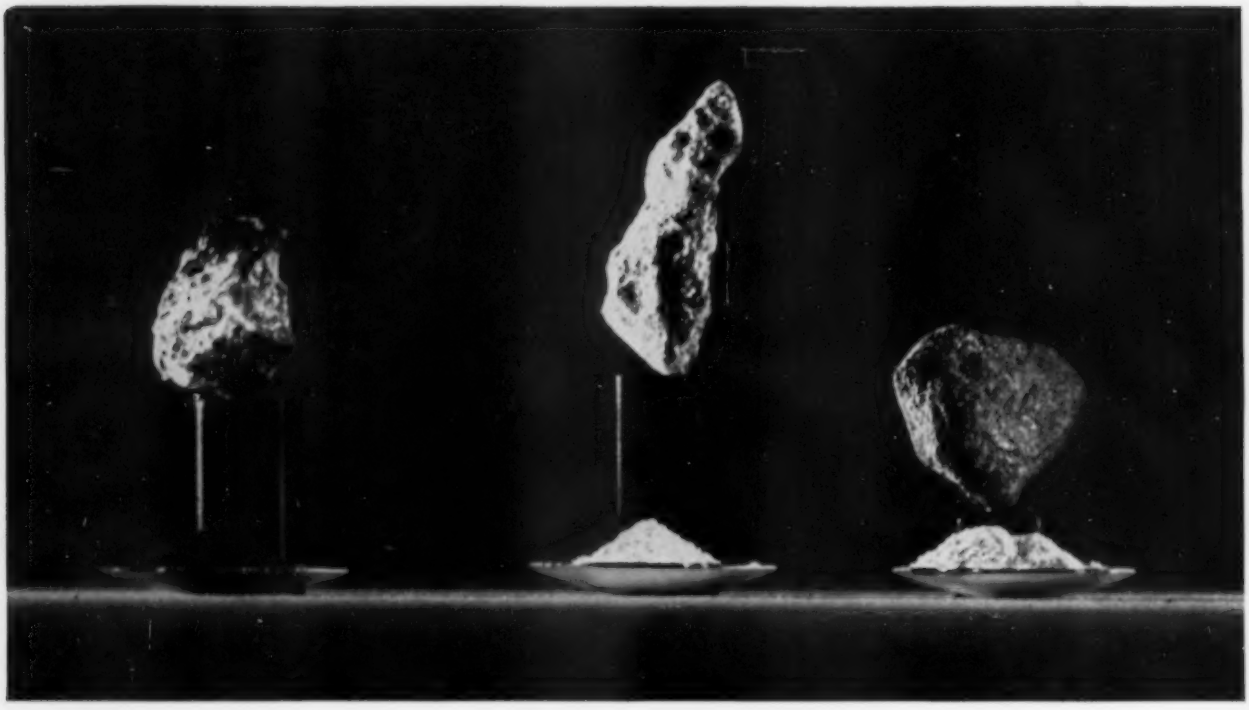

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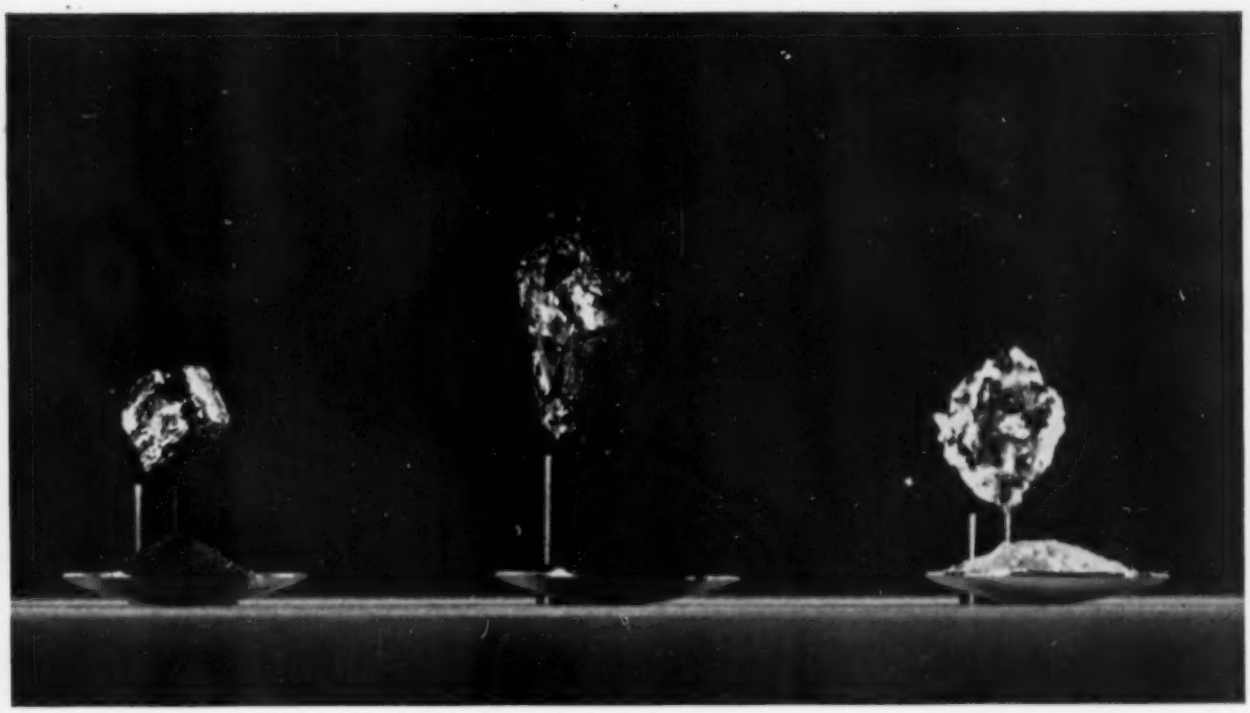

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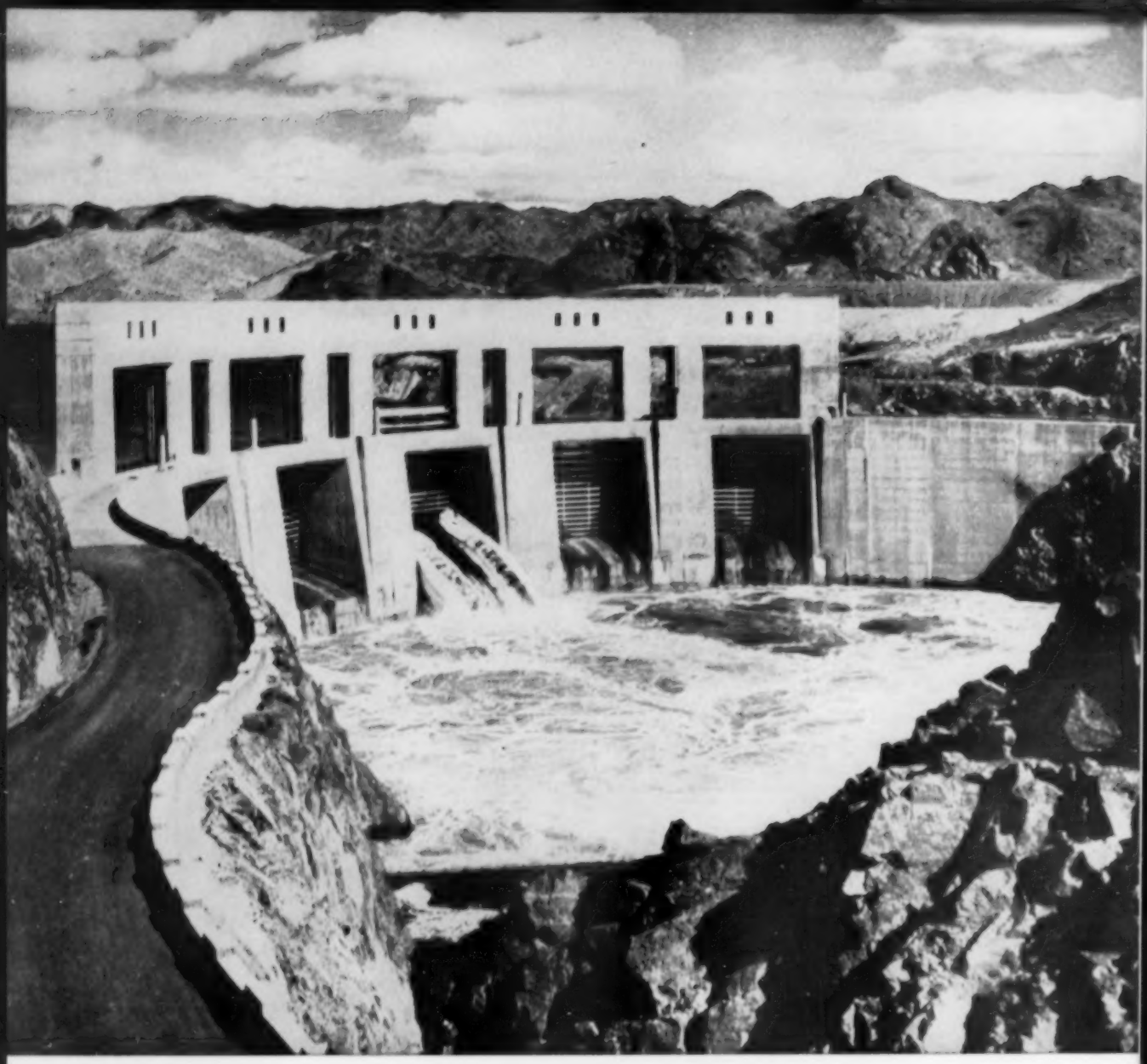

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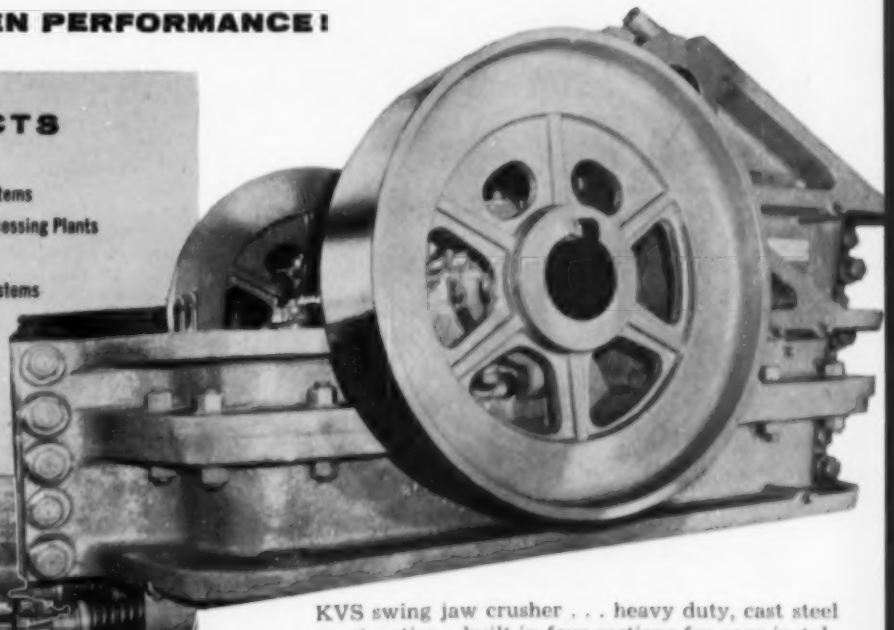

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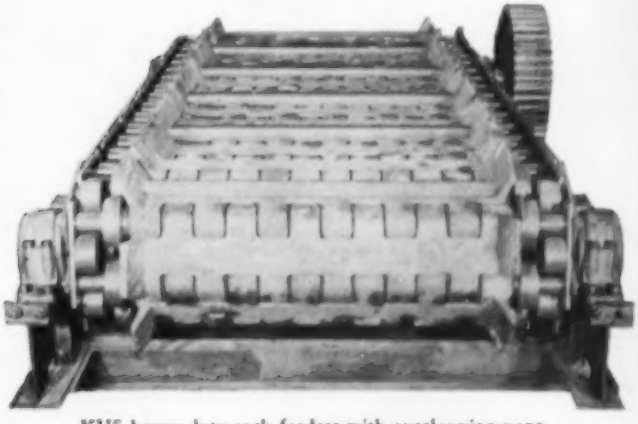

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ASME Handbook: Engineering Tables, edited by Jesse Huckert, *McGraw Hill Book Co., Inc.*, $12.00, 704 pp., 1956.—This reference book, sponsored by the American Society of Mechanical Engineers, is one of four which comprise the complete **ASME Handbook**. It contains many tables of design standards not generally found in handbooks, such as the table of involute functions, methods for finding loads on ball bearings and information on bars and tubes suitable for reworking into gears, levers, shafts or screws. Conveniently arranged in 15 sections the book covers such subjects as bar stock and shafting; bearings; spur, helical and herringbone, bevel, and worm gears; keys; bolts; springs; aircraft and mechanical tubing; and electrical motors. The author, who is professor of mechanical engineering at Ohio State University, was an editor of *Product Engineering*.

Please Order These Publications from the Publishers

Uranium Country, by Kathleen Bruyn, *University of Colorado Press*, Boulder, Colo., $2.00, 165 pp., 1955.—A history of uranium mining in America, this book traces the story back to the search for radioactive ores in Colorado, from the discovery of pitchblende in 1871, to the modern bonanzas of Vernon Pick. The facts were gleaned from the men who made and are still making uranium history. Miners, prospectors, manufacturers and financiers, as well as famous scientists such as Marie Curie are all characters in the book. With emphasis on the orebodies, the theme is that most of the great discoveries of the present uranium era are actually re-discoveries of abandoned claims or of areas known to have produced mineral deposits years ago. Ore strikes in virgin territory have been accorded only passing attention. Methods of ore processing, theories of ore deposition, and modern techniques of prospecting are discussed.

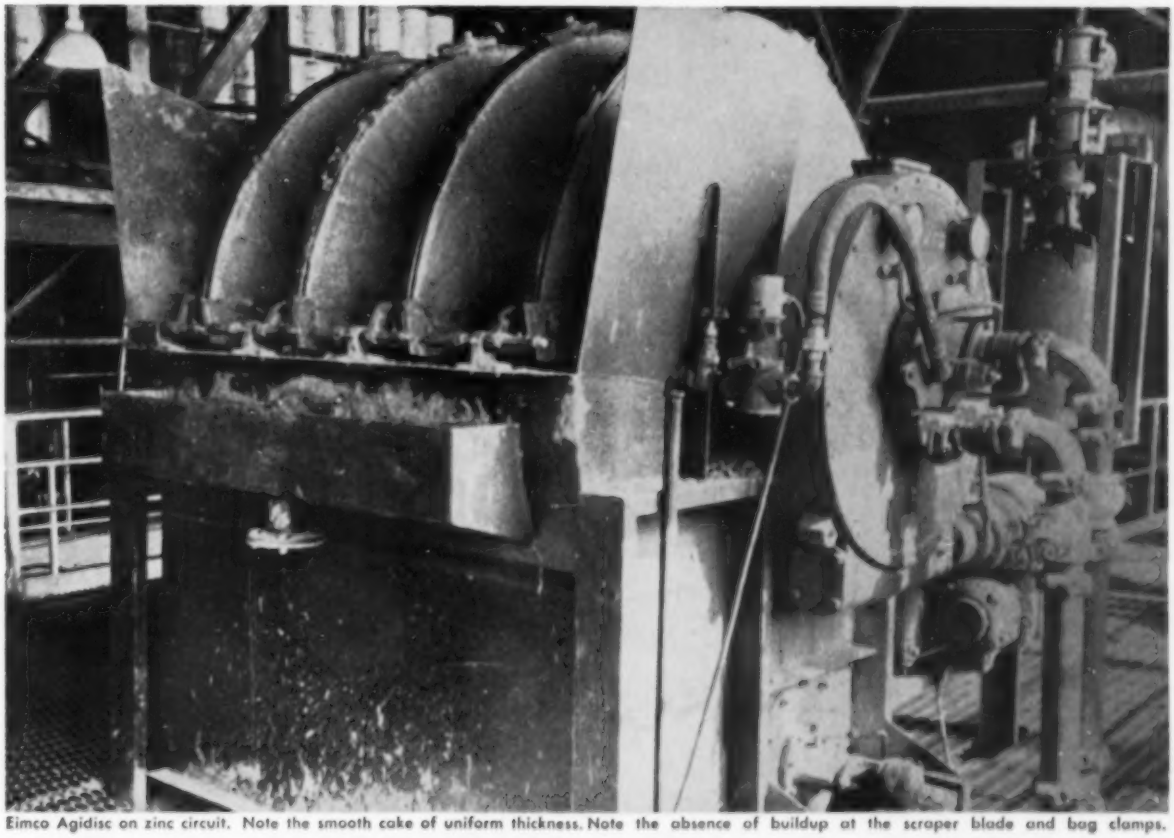
The Mining Journal Annual Review, *The Mining Journal Ltd.*, 15 Wilson St., Moorgate, London, E. C. 2, England, $1.00 postpaid, 324 pp., 1956.—World events in mining in 1955. More than 100 articles on technical advances, finance, metals, and progress reports on the British Commonwealth's principal mining companies, as well as on other mining and investment companies.

Lehrbuch der Bergbaukunde, vol I, by C. Hellmut Fritzshe, *Springer-Verlag*, Berlin, available in the U. S. from *Stechert-Hafner Inc.*, 31 E. 10th St., N. Y., approx. $8.50, 759 pp., 9th edition, 1955.—This first volume of a series on mining and mining methods provides a comprehensive and extensive treatment of drilling, recovery methods, haulage, ventilation, dust problems and lighting. Although the emphasis is on coal mining, other types of underground mining—ore, salt, petroleum—have been considered. Fully illustrated, the book is encyclopedic in character.

Mineral Resources of the Navajo-Hopi Reservations in Arizona and Utah, vol II, Nonmetallic Minerals, by George A. Kiersch, *University of Arizona Press*, Tucson, $1.00, 105 pp., 1955.—Prepared for the Bureau of Indian Affairs, this book treats the geology, evaluation and uses of nonmetallic minerals with reference to the rocks of the area, landforms, limestone and clay materials. Also detailed are kaolin, coal, gypsum, puzzolan, semi-precious stones and special sand. The book contains two appendices and a glossary as well as maps, charts, tables, and photos.

Geology of the Late Paleozoic Horseshoe Atoll in West Texas, by D. A. Myers, P. T. Stafford, and Robert J. Burnside, *Bureau of Economic Geology, University of Texas*, Austin, Texas, Pub. No. 5607, $2.00, 113 pp., 1956.—Produced by the combined efforts of the U. S. Geological Survey and the Bureau of Economic Geology, the book treats the characteristics of reef rocks, stratigraphy of rocks beneath, in, and overlaying the atoll. The Horshshoe atoll is a subsurface accumulation of fossiliferous limestone, 3000 feet thick, which was deposited during Pennsylvanian and early Permian time in the northern part of western Texas. Large quantities of petroleum have been obtained from the eastern and southern parts of the atoll. The book contains large maps in color.

Mineral Facts and Problems, U. S. Bureau of Mines Bulletin 556, by various authors, Superintendent of Documents, *U. S. Government Printing Office*, Washington 25, D. C., $5.75, 1048 pp., 1956.—Past and present of over 85 minerals are presented in an authoritative study that also forecasts possible mineral futures. Intended as a reference work for researchers, businessmen, planning authorities, investors, the encyclopedia is written for nontechnical as well as technical readers. Individual chapters on specific minerals are available at prices ranging from 5 to 40 cents each.



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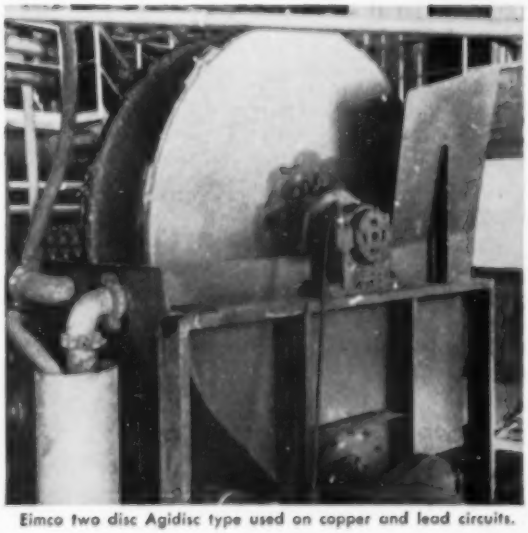

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BOOKS

(Continued from page 668)

Geology of the Late Paleozoic Horseshoe Atoll in West Texas, by D. A. Myers, P. T. Stafford, and Robert J. Burnside, *Bureau of Economic Geology, University of Texas,* Austin, Texas, Pub. No. 5607, $2.00, 113 pp., 1956.—Produced by the combined efforts of the U. S. Geological Survey and the Bureau of Economic Geology, the book treats the characteristics of reef rocks, stratigraphy of rocks beneath, in, and overlaying the atoll. The Horseshoe atoll is a subsurface accumulation of fossiliferous limestone, 3000 feet thick which was deposited during Pennsylvanian and early Permian time in the northern part of western Texas. Large quantities of petroleum have been obtained from the eastern and southern parts of the atoll. The book contains several large maps in color.

Metal Statistics 1955, edited by N. J. Langer, *American Metal Market,* 18 Cliff St., N.Y.C., $3.50, 864 pp., 1955. —This 48th annual edition contains a general assortment of statistical information on ferrous and nonferrous metals. Included are new statistical tables such as annual shipment of products by domestic copper and brass mills; primary aluminum producing capacities of the U. S. and Canada; consumption of nickel in the U. S. by uses; annual production, shipments and stocks of brass and bronze ingots in the U. S. etc. Statistics dealing with production, consumption, imports, and exports are gathered from such authentic sources as the AISI, USBM, Census Bureau. For the first time, the Machol Edge Index is included.

American Universities and Colleges, 7th edition, edited by Mary Irwin, *American Council on Education,* 1785 Mass. Ave., Wash. 6, D. C., $12.00, 1210 pp., 1956.—A classic reference for student counselors, college administrators, researchers, and writers, this book provides full data on 969 accredited colleges in the U. S., Alaska, Hawaii and Puerto Rico. Also included is data on 2016 approved professional schools in 23 fields.

SKINNER COMINCO ROASTER

HANDLING 300 T/24 HRS. ZINC SULPHIDE CONCENTRATES AT EAGLE-PICHER

This CIW Skinner type Cominco Suspension Roaster is operating in The Eagle-Picher Co. plant at Galena, Kansas. It is 22' 9" I.D., has two dryer hearths, a calcine collecting hearth, and a 36' high combustion chamber. This unit incorporates features of the well known CIW Skinner Multiple Hearth Roaster and is regularly producing low sulphide sulphur in the calcine and high sulphur dioxide concentrations in the exhaust gases.

The Skinner modification of the Cominco Roaster is manufactured by CIW and was designed in cooperation with and built for The Consolidated Mining and Smelting Company of Canada Limited, licensors of the Cominco Roaster and Suspension Roasting Process. In addition to the roaster shown here, several of these units are being successfully operated by The Consolidated Mining and Smelting Company of Canada Limited and the Societe Anonyme des Mines et Fonderies de Zinc de la Vieille Montagne.

Photo shows top half of the CIW Skinner type Cominco Roaster at Eagle-Picher.

Control panel for roaster

YOU CAN HAVE THE ADVANTAGE OF CIW's 38 YEARS ROASTING EXPERIENCE . . . ***JUST CALL, WIRE OR WRITE***

COLORADO IRON WORKS CO.

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A SUBSIDIARY OF THE MINE & SMELTER SUPPLY CO.

LETTERS

Dear Sir:

You have heard much lately about the shortage of mining engineering graduates. However, a few brief statistics on the 54 companies and organizations contacting us for graduates during the past year may be of interest to you.

Following is a functional classification by type of this group:

Coal Mining	12
Hard Rock Mining	15
Oil and Gas	9
Manufacturing	6
Government	6
Others	6
Total	54

The number of companies in each category does not tell the complete story as some may have hired the entire class of seven graduates. Also, many steel companies interview for more than one type of work.

It is estimated conservatively that at least ten job opportunities exist for each graduate mining engineer. In order to improve the situation, we are recommending industry supported programs for qualified high school graduates.

Attached is an announcement which is typical of the method used to publicize these awards. Also enclosed is an illustrated folder on mining engineering for your information.

A. W. ASMAN, Head
Department of Mining
Pennsylvania State
University

Eimco 630 Tractor-Excavator is air powered for tunnel operation.

EIMCO 630 EXCAVATORS SPEED DEVELOPMENT

High speed development can be maintained underground with only the normal shift crew. The Eimco 630 helps the miner or contractor realize this goal because it operates faster and requires no track.

The Eimco 630 Excavator is mounted on the Eimco 630 air or electric (A.C.) powered tractor and the combination unit provides the operator with a really mobile and versatile loading machine.

With its independently controlled tracks the Eimco 630 Excavator can dig in at the toe of the muck pile, bulldoze out into stations cut along the side, walk over, around and through unworkable areas in the mine eliminating the necessity for laying rails.

It is also in use for shaft sinking in round and rectangular shafts. It is being used on slopes and for mucking out sub-level ore pockets.

In all of the applications where the 630 has been employed the costs of handling the material have been less than with previously used equipment.

The Eimco 630 is rugged, requires a minimum of maintenance attention and has the same dependability that has been characteristic in Eimco made equipment for more than half a century.

Write for complete information.

THE EIMCO CORPORATION

Salt Lake City, Utah—U.S.A. • Export Offices: Eimco Bldg., 52 South St., New York City

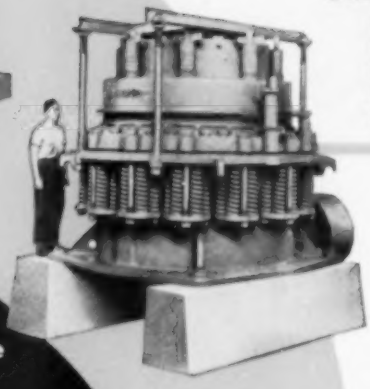
New York, N. Y. Chicago, Ill. San Francisco, Calif. El Paso, Tex. Birmingham, Ala. Duluth, Minn. Kellogg, Ida. Baltimore, Md. Pittsburgh, Pa. Seattle, Wash. Pasadena, Calif. Houston, Texas Vancouver, B. C. London, England Gateshead, England Paris, France Milan, Italy Johannesburg, South Africa

SYMONS® PRIMARY GYRATORY CRUSHERS

Built in 30", 42", 54", 60" and 72" sizes, for capacities up to 3500 or more tons per hour.

SYMONS CONE CRUSHERS

Built in both Standard and Short Head types, in sizes from 22" to 7', in capacities from 6 to 900 or more tons per hour.



NORDBERG MACHINERY

for URANIUM PRODUCTION

keeps pace with the Atomic Age

URANIUM—the Wonder Metal of the Atomic Age—is today one of the most sought after elements in the earth's crust. And with good reason when you consider that one pound of pure Uranium will produce by fission as much energy as the combustion of 20,000,000 pounds of coal. This powerful new servant serves mankind in many ways and each day its benefits grow increasingly greater.

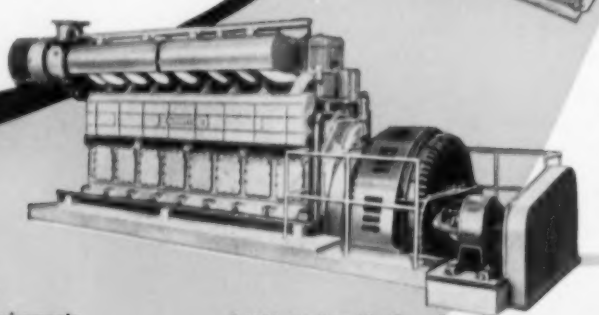
Increasing too is the role being played by Nordberg Machinery in the efficient processing required for production of this valuable metal. From the Congo and South Africa to the ore bodies of the United States and Canada, leading producers are installing Symons Gyratory Crushers for primary breaking, Symons Cone Crushers for finer reduction, Symons Grizzlies and Screens for scalping and sizing, and Nordberg engines for mine and mill power.

SYMONS VIBRATING BAR GRIZZLIES

Built for heavy duty large capacity scalping service. Particularly effective when handling wet, sticky or gummy ores. Will handle feeds up to 30" and larger.

SYMONS ROD DECK SCREENS

Highly efficient, big capacity units adaptable to mill feed screening and for handling heavy feeds of wet, sticky ores.



NORDBERG ENGINES

Built in a wide range of sizes from 10 to over 12,000 horsepower, including Diesel, Duafuel® and Spark-Ignition Gas types for low cost power generation.

Write for information on your machinery requirements.

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C-456

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Milwaukee, Wisconsin

SYMONS . . . A Registered Nordberg Trademark known throughout the world.

NORDBERG

MACHINERY FOR PROCESSING ORES and INDUSTRIAL MINERALS

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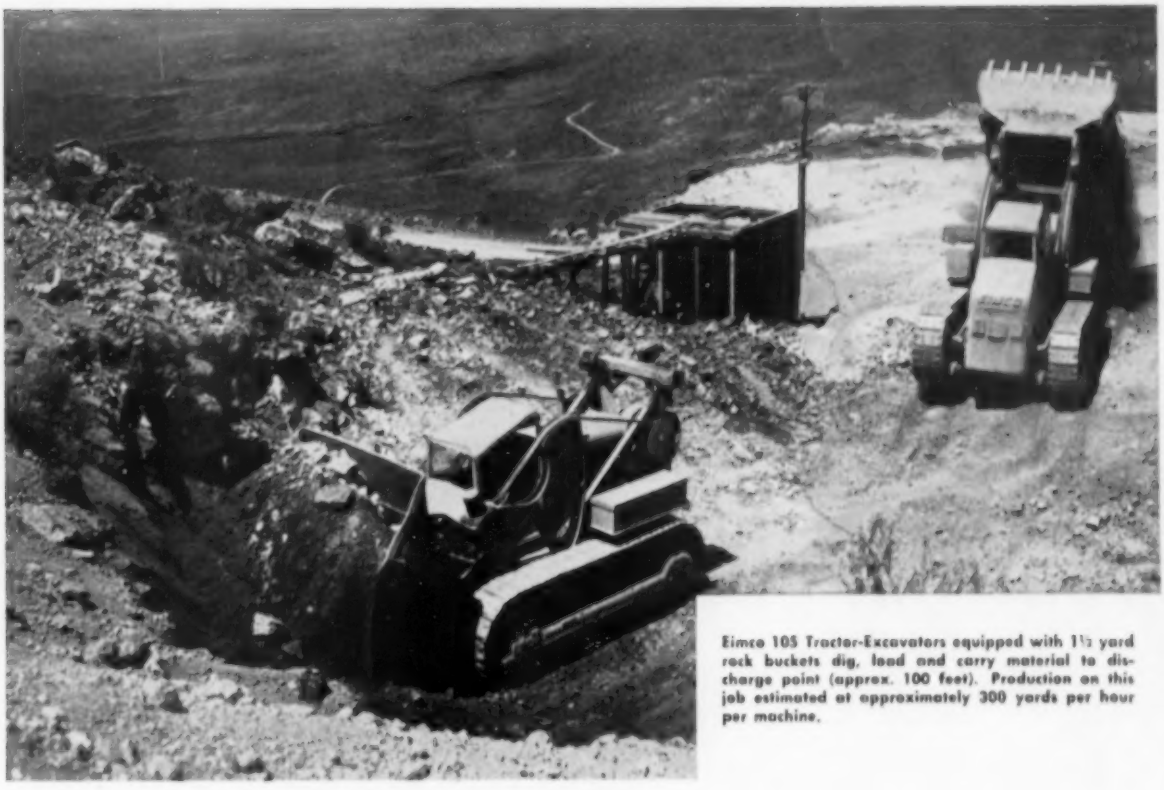


Eimco 105 Tractor-Excavators equipped with 1½ yard rock buckets dig, load and carry material to discharge point (approx. 100 feet). Production on this job estimated at approximately 300 yards per hour per machine.

SPEED, VERSATILITY AND DEPENDABILITY WITH EIMCO 105 TRACTOR-EXCAVATORS

Eimco 105 Tractor - Excavators, shown above, dig and load more rock at less cost per yard than any other earthmoving machine.

These Eimco Excavators are multi-purpose machines. They are easily and quickly moved from one spot to another where work is needed. They are used as prime movers, on production excavation and loading jobs, for bulldozing and other work around the plant.

In production work, the Eimco 105 Tractor-Excavator will load in rock at the rate of 5-6 yards per minute and in easier materials such as sand and gravel, the 105 will load as much as 8-9 yards per minute.

Eimco 105's do a better job than the conventional tractor mounted machine because of Eimco's exclusive operating features. Independent track operation increases the effectiveness of the machine in speed of maneuverability, saves travel time and wear and tear on track assemblies, chains and all crawler parts.

The Eimco power shift saves wear and tear on operators, keeps them fresh and alert all day long. Operator position up front makes it possible to see what is being done.

All cast-alloy steel parts are standard equipment on Eimcos at no extra cost. Torque converters, also standard, provide better engine operating flexibility and protect the engine from shock.

Eimco 105's are different in construction, design and operation. They are the only really modern tractors in production today. An Eimco engineer will be glad to tell you about the 105 and answer your questions. When you need a Tractor or Tractor-Excavator be sure to see an Eimco 105 at work, and talk to people using them. You are invited to compare the Eimco 105 Tractor, piece for piece with the tractor you are using or any other tractor.

THE EIMCO CORPORATION

Salt Lake City, Utah—U.S.A. • Export Offices: Eimco Bldg., 52 South St., New York City

New York, N. Y. Chicago, Ill. San Francisco, Calif. El Paso, Tex. Birmingham, Ala. Duluth, Minn. Kellogg, Ida. Baltimore, Md. Pittsburgh, Pa. Seattle, Wash. Pasadena, Calif. Houston, Texas Vancouver, B. C. London, England Gateshead, England Paris, France Milan, Italy Johannesburg, South Africa

EIMCO

B-807

POWERHOUSE ...one D397 does it all!

In one of the world's largest phosphate beds, near Soda Springs, Idaho, this CAT* D397 Electric Set is the main source of power for the crushing plant shown here. Output of the crusher, owned by Morrison-Knudsen Co., Inc., is 1000 tons per eight-hour day of 1½"—⅜" quartzite. The crushed quartzite is raw material used in Monsanto Chemical Company's nearby elemental phosphorus plant.

The D397 Electric Set furnishes power for a 125 HP motor on the Simmons cone crusher, a 100 HP motor on the Cedarapids jaw crusher, and nine motors totaling 172½ HP on conveyors and vibrator screens.

Now there is a new Caterpillar D397 Electric Set with Turbocharger, developing 350 KW. It is self-regulated for easy, adjustment-free operation and steady voltage. It incorporates Caterpillar's amazing new generator, which exactly matches the D397 Engine powering it.

Reliability and long life are important in an electric set, which is often called upon for continuous operation. The Caterpillar D397 has both, with such features as aluminum alloy bearings, "Hi-Electro" hardened crankshaft and cylinder liners. It also has highly effective oil and air filters designed to protect the engine's Caterpillar-built precision under even the dustiest conditions.

There is a complete line of Caterpillar portable and stationary Electric Sets, up to 350 KW. Your Caterpillar Dealer — who provides skilled service and parts you can trust — has full details on the engine or electric set best suited to *your* operation. See him today.

Caterpillar Tractor Co., Peoria, Illinois, U. S. A.

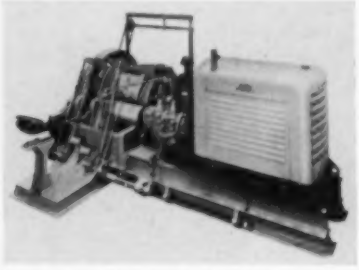
CATERPILLAR*

*Caterpillar and Cat are Registered Trademarks of Caterpillar Tractor Co.

MODERN HEAVY-DUTY ELECTRIC SETS

Mine Hoists

A new series of Vulcan-Denver hoists for small vertical and incline shaft operations features fluid coupling drive. The gasoline powered hoists by *Vulcan Iron Works Co.*



have double post brakes, and a spring mounted operator seat. Model GFT-40 shown above has a rated pull of 4000 lb and is used with 1500 ft of ⅝ in. rope. **Circle No. 1.**

Overhead Clothes Storage

Dark corners and congested aisles of conventional locker rooms can be eliminated through the use of overhead storage. *Moore Co.* has the wire Lockerbasket which holds a change of clothes and equipment and is suspended from change room ceilings to keep clothes fresh and dry. Baskets are raised and lowered by individual chains and pulleys and each is provided with a tamperproof locking assembly. **Circle No. 2.**

Tractor-Mounted Backhoe

Designed for the Caterpillar D4 tractor, the *Hyster Co.* hydraulic backhoe can dig down to 13 ft and

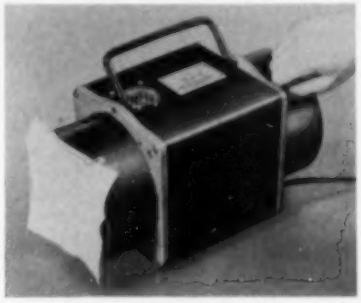
has a capacity of ½ cu yd. Stabilizing outriggers are hydraulically operated, rack and gear swing mechanism turns at uniform speed through a 240° arc, and dippers are offered in cutting widths of 13, 21, or 29 in. **Circle No. 3.**

Power-Rotated Jib Crane

Available in capacities of 6, 7½, 10, and 15 tons, a power-rotated jib crane by *R. G. LeTourneau Inc.* has a 25-ft boom which is push-button operated. Positive inching control is provided by electro-mechanical motor brakes. One man can operate the crane which may be used for outdoor or indoor work and expensive tie-ups of overhead cranes can be avoided. **Circle No. 4.**

Air Sampler

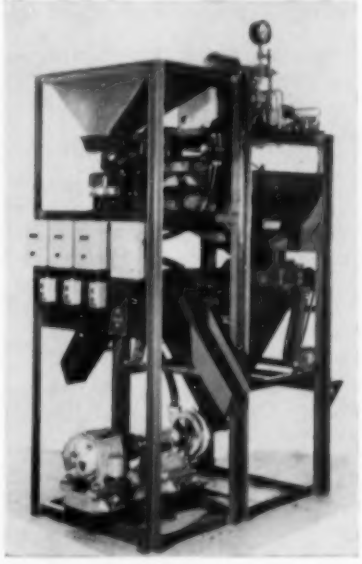
Mine Safety Appliances Co. has the Fixt-Flo air sampler designed to collect aerosol contaminant samples for analysis in connection with hygiene problems while automatically maintaining a constant collection flow. A damping adjustment feature



compensates for resistance to air flow that increases as material collects on the filter. Increased accuracy in determining volume is assured in each of three selected flow rates: 15, 30, and 50 cfm. Sampler measures 7¼x7¼x15 in. and weighs 13.5 lb. **Circle No. 5.**

Laboratory Separator

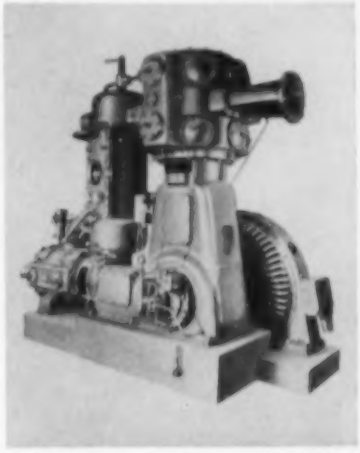
The Ore & Chemical Corp. has announced a laboratory-size heavy-media separatory vessel designed for batch-type testing with feed capacities up to 1000 lb per hr. Simplicity



and high ratio of productive pool area are claimed for handling feeds in size range −1 in. to +10 mesh. **Circle No. 6.**

Heavy Duty Compressors

Atlas Copco Pacific Inc. has increased its line to include the Atlas Copco AR series of compressors. Especially designed for long-range mining operations, the six models available range in size from 330 to 3220 cfm. Steel skid mounts are



available on smaller units in the series and they may be moved as work progresses. Low operating speeds, less operating heat, and less power requirement to drive are claimed for this Swedish-made equipment. **Circle No. 7.**

Self-Propelled Scrapers

Several design changes have been made in *International Harvester Co.* Payscrapers. Capacity of the 262-hp model 75 is increased to 20 cu yd heaped and the 172-hp model 55 to 14 cu yd heaped. Other changes include: straight-back bowl, lower

draft frame, increased visibility, higher apron lift, larger push block, and improved ejector. Model 75 also features a cerametallic clutch claimed to give positive engagement, reduced slippage, less frequent adjustment, and longer life. **Circle No. 8.**

IN LATTER-DAY GOLD RUSH, THIS **D8** WORKS **FAST**

When each yard of dirt contains only a few cents' worth of gold, and when you have to pack a full year's mining into three summer months—you need dependable equipment that works *fast!*

That's why B. Bratsberg has a Caterpillar D8 Tractor. He owns and operates Gold Bottom Placers near Dawson, Yukon Territory. His mine is two miles from "Henderson's Discovery," which started the gold rush in 1896. But his mining method is vastly different from 60 years ago: Mr. Bratsberg's D8 strips and 'dozes into the sluice box and stacks tailings. With its No. 8A Bulldozer it moves 4½ cu. yd. each load and averages a load a minute while sluicing.

"Our short season means we've got to keep going," Mr. Bratsberg says. "I wanted a tractor that was dependable and able to do a lot of hard work. My D8 moves a lot of yardage economically." During the three-month summer season this CAT* D8 Tractor works nine hours a day, seven days a week. It went seven years (4800 hours) before overhaul.

And now there is a new D8 that will perform even better. The Caterpillar D8 Tractor, with torque converter or with exclusive oil clutch, has a completely new, 191 HP diesel engine. Controls are hydraulically boosted and easy on the operator. "Live-shaft" drive lets you operate rear-mounted equipment independent of the flywheel clutch.

The new D8 is available with a choice of bulldozer blades and controls to suit your operation. Your Caterpillar Dealer will be glad to demonstrate the D8 or other Cat Diesel Tractor on *your* job. Call him today and find out about the tractor that will produce the most at the lowest cost.

Caterpillar Tractor Co., Peoria, Illinois, U.S.A.

CATERPILLAR*

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NAME THE DATE... YOUR DEALER WILL DEMONSTRATE

(21) COAL BLASTING: Explosives Dept. of *American Cyanamid Co.* has compiled a reference bulletin on the use of permissible explosives, listing those available for all shooting conditions in coal mining. Testing, preparation and blasting procedure, and storage methods are detailed in the 12-page booklet.

(22) GOLD FLOTATION: Bulletin M4-B85 from *Denver Eqpt. Co.* describes gold recovery by flotation and cyanidation at the LaLuz mines in Nicaragua. Geology of the ore body, milling practice, and a flowsheet and operating costs of the operations are included.

(23) PRESSURE FILTRATION: *Graver Water Conditioning Co.* has a 12-page bulletin on the uses, design features, and engineering details of their pressure sand and gravel filters. Graver filters provide a means of delivering filtered water without the necessity of repumping after filtration. Brochure has cutaway views of horizontal and vertical filters and details of strainers, distributor systems, and piping arrangements.

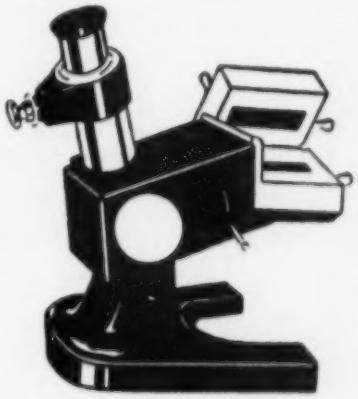
(24) JAW CRUSHER: Believed the largest overhead eccentric jaw crusher in current production, the 4448 WRB by *Universal Engineering Corp.* features a stress-relieved, welded-steel base; hydraulically removed spherical self-aligning bearings; rocker-type steel toggle plate, steel-plated shims and hydraulic cylinders. Approximate capacities range from 200 to 950 tph.

(25) INDUSTRIAL PRESSES: Manley Div. of *American Chain & Cable Co.* has a line of hydraulic and air-operated industrial presses of 25 to 150 ton capacity, and accessories. Detailed information is included in 16-page bulletin DH-486 on these items and on their 3-ton capacity arbor presses, the 2-ton capacity mobile floor crane, mobile floor trestles, and a 2-ton capacity universal floor crane.

Free Literature

(26) SALT BATH FURNACES: *Hevi Duty Electric Co.* has the Bellis line of salt bath furnaces which feature immersed alloy electrodes with water-cooled terminals to prevent conduction of heat to the transformer; interchangeable ceramic, steel, or alloy pots; sliding cover; and low operating voltage. Bulletin 655 gives complete information.

(27) LAB INSTRUMENTS: A line of European-made laboratory instruments distributed by *C. A. Brinkman & Co.* is shown in an assorted group of folders. Listings cover models of balances, motion picture assemblies, coulometers, microscopes, manipulators, dispensers, vacuum



driers by a number of foreign manufacturers. Sartorius' Projecta analytical projection balance, for example, has all-metal housing, circular doors, table level dial for built-in weights, single air damper, swivel type weight compartment in base.

(28) SCREEN ENCLOSURE: Dust-tight enclosures for floor-mounted vibrating screens by *Allis-Chalmers*, described in bulletin 07B8423, are available for the model AVS and model S. Hinged top cover, swing-up discharge spout, and removable side panels are featured. High pressure fittings outside the enclosure permit simple lubrication of vibration mechanism bearings.

(29) PROTECTIVE FABRIC: A combination of nylon and plastic, Herculite is a protective fabric of light weight said to be ten times stronger than canvas and bidding to overtake canvas for industrial uses. *Herculite Protective Fabrics Inc.* claims the material will not support combustion, is waterproof and not affected by acids, grease, oil, or the corrosive effects of salt water. Material is shrink-proof and may be easily repaired if punctured.

(30) LIGHT METAL TUBING: Titeflex 500 is a metal tubing formed by winding a thin metal strip into a tube of desired diameter with smooth bore inside and helical ridges outside. *Titeflex Inc.* says that the tubing is especially useful for ducts in construction work. Steel strip is lead-coated for certain applications, can be cut with a knife, and may be hand-formed for complex layouts.

(31) TRIP LAMP: New Edison Four-Cell Trip Lamp is designed for greater visibility and dependability. Battery has a rating of 4.85 v at 1 amp and may be recharged on standard equipment. Lamp is rectangular in shape and may be hung by its forged steel hook to the back or side of any mine car. Bulletin 0201-2 from *Mine Safety Appliances Co.* describes the complete Edison Electric Lamp line.

(32) ROLLER CHAIN: Book 2457 from *Link Belt Co.* has 148 pages on their line of roller chain with sections on stock drives, casings, sprocket wheels, installation and maintenance, lubrication.

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

7

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after Oct. 15, 1956—if mailed in U. S. or Canada

Please send { More Information ☐ / Price Data ☐ / Free Literature ☐ } on items circled.

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Company _____

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City and Zone _____ State _____

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
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51	52	53	54	55	56	57	58	59	60
61	62	63	64	Students should write direct to manufacturer.					

(33) RESIN-IMPREGNATED MATERIALS: Celeron is a synthetic resin impregnated material that is said to be tough, resilient, mechanically strong. Manufactured by Continental-Diamond Fibre Div. of *Budd Co.*, Celeron is available in four forms: sheets, molded-laminated material, molded macerated material, and combination laminated-macerated material.

(34) REAR DUMP: An 8-page catalog from Euclid Div. of *General Motors Corp.* describes the R-15 Rear-Dump unit. Rated payload is 15 tons with standard tires and 18 tons with optional larger tires. Power is supplied by a 218-hp diesel engine, and clutch is air-assisted. A quarry body with flared sides for rock hauling is optional.

(35) CONVECTION FURNACES: Temperite Convection Furnaces use coiled nickel-chromium wire elements held in ceramic refractories to yield temperatures to 1350°F. *Hevi Duty Electric Co.* makes nine standard sizes of the furnace which features mounting of heat units on the sides of the work chamber and graded shell insulation to lessen heat loss.

(36) STEEL SHELVING: Tips on planning shelving installations are included in a catalog of steel shelving materials by the Hallowell Div. of the *Standard Pressed Steel Co.* Photos, drawings, suggested floor plans, and a shelf-capacity chart make the booklet a useful guide to planned storage.

(37) SELF-PROPELLED SCRAPERS: Euclid Div. of *General Motors Corp.* offers three models in their overhung engine line of scrapers, models S-7, S-12, and S-18. Condensed specifications provide data on each scraper in separate brochures: Catalog #505 covers the S-7 which has a payload of 7 cu yd struck; #506 describes the S-12, of 12 cu yd capacity; and #507 the S-18 which carries 18 cu yd struck. Hydraulic level action of apron, bowl, and ejector provides independent control of all scraper operations.

(38) FLOTATION: "Mineral Flotation with Armour Cationic Chemicals" is a new booklet by the Chemical Div. of *Armour & Co.* that describes the properties, application, and advantages of cationic chemicals in flotation. Ore dressers, flotation men, and beneficiation engineers will find concise information on principle, chemicals used, conditions needed, and inherent advantages in the use of this flotation method. A listing of technical services and available literature supplements the data.

(39) SPECTROCHEMICAL ANALYSIS: A colored brochure from *National Spectrographic Sales Corp.* illustrates the scope of the science of spectroscopy. From sample preparation to line spectra recording and interpretation, the steps taken in the spectrographic laboratory are shown. Features of the Spec-Trol laboratory are also outlined.

(40) WELDED TUBES: A 6-page folder containing data on the design, maintenance, and repair of condensers and heat exchangers is offered by Tubular Products Div. of the *Babcock & Wilcox Co.* Bulletin points up the advantages of electric resistance welded carbon steel tubing in heat transfer uses. Uniform ductility and dimensional accuracy are claimed for B&W pressure tubes.

(41) GENERATING PLANTS: The Onan electric generating plants being used in mobile, emergency standby, and standard service in many applications are detailed in a new 8-page catalog by *D. W. Onan & Sons Inc.* Air-cooled and water-cooled, gasoline and diesel models are shown in the listings in various types and sizes. A detailed "Model Selection Guide" for each type aids in finding the most suitable unit.

(42) SUB-ZERO TEMPERATURE CABINET: Features of the *Webber Engineering Corp.* subzero temperature cabinet are found in data sheet 61. Cabinet has 2 cu ft capacity, front or top opening, and is used for testing of electronic components, cold treatment of metals, thermal contraction, and expansion fitting. With an ambient range from (110°F) to —140°F, the temperature is adjustable from 150°F to —150°F.

(43) "JEEP" INDUSTRIAL ENGINES: A 6-page folder from *Willys Motors Inc.* details their line of "Jeep" 4-cyl industrial engines and power units. Lightweight pistons of tin-coated aluminum alloy, with hard-chrome plated top compression rings; positive crankcase ventilation; drop forged steel crankshafts, statically and dynamically balanced and counterweighted are some features of these engines that are rated at 20 to 55 hp.

(44) HYGIENE HISTORY: "History of Industrial Hygiene Foundation" is a bulletin marking the twentieth year of active service of that organization. Member companies are assisted in the solution of specific industrial health problems and in the following of medical programs in industry.

(45) WELDING PRODUCTS: Stock sizes and prices of Tisco Timang manganese-nickel-steel welding products are provided in 4-page bulletin 3-56-1M from Taylor-Wharton Div. of *Harsco Corp.* Included are rods, hot-rolled plates, special shapes, wedge bars, tooth repointers, and applicator bars.

(46) SURFACING STEEL: *Chromalloy Corp.* has an 8-page bulletin, "Chromallizing Improves Surface Properties of Steel" that describes the Chromalloy process for increasing the heat, wear, and corrosion resistance of steel by high temperature diffusion of chromium into the surface. A revision of a previous folder, it discusses advantages, costs, and applications.

(47) NICKEL TUBING: Catalog 12 from *Superior Tube Co.* has a general discussion of nickel and nickel alloys and a table of the specific mechanical properties and chemical compositions of 13 kinds of tubing drawn at their mill. Described tubing is supplied in standard outer diameters from 0.012 to 1.125 in. Table of standard production limits includes maximum and minimum walls for tubes in each standard size.

(48) ELECTRODES: Characteristics, properties, and sizes of the five grades of Ampco-Trode bronze electrodes are available in bulletin W-25a by the Weldrod Dept. of *Ampco Metal Inc.* Revised booklet is intended as a guide to maintenance and repair of products serviced by bronze electrodes.

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MINING ENGINEERING

29 WEST 39th STREET

NEW YORK 18, N. Y.

New Thor all-purpose one-man drilling machine

THOR introduces the powerful new 390 drilling machine with a host of features which speed up drilling operations and reduce operator fatigue.

This Thor drill is equipped with heavy-duty aluminum cylinder and chrome-plated steel piston rod and a specially designed 45 lb. sinker rock drill with integral feed leg connection. It can be easily converted for sinking. Thor 390 is equipped with a line oiler and automatic water valve which can be replaced with a plug for dry operation or collaring.

All controls are grouped on the backhead for safe convenient operation. A six position throttle, a Thor exclusive, gives a complete range of control from complete shut-off of air and water, to full air and water supply in logical steps for easy convenience and instant control.

Optional feed travel legs available in 36", 48" and the new model 680 72" telescopic leg.

Your nearest Thor representative will be glad to arrange for a demonstration. Thor Power Tool Co., Aurora, Ill.

MODEL 390
DRILLING MACHINE
MOUNTED ON 72"
680 TELESCOPIC LEG

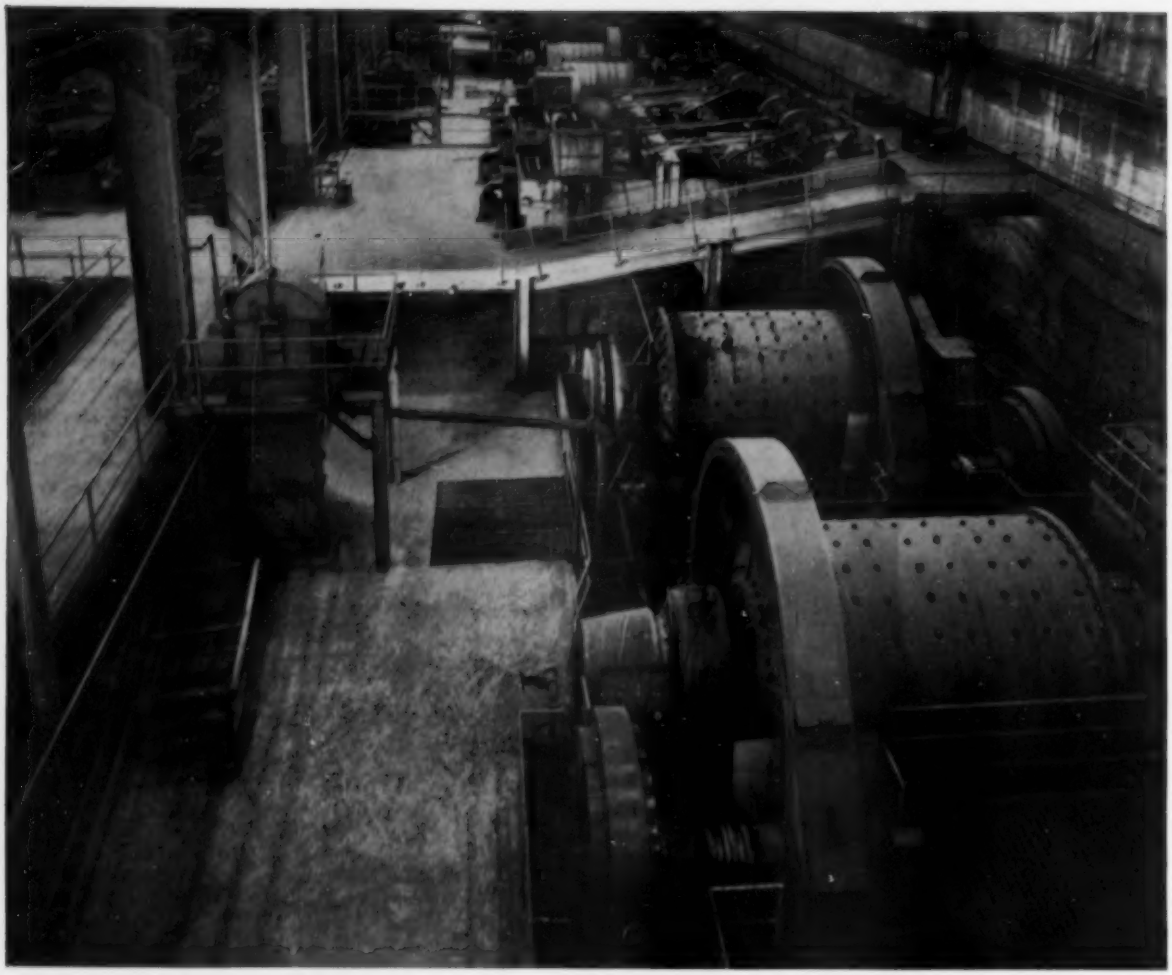
Thor
PORTABLE POWER
TOOLS

New Thor 390—drifter, sinker, stoper—all in one

The Thor No. 390 is easily adaptable for all types of drilling at all angles. The drill can be removed from the leg for use as a sinker by loosening only one nut. Tool swivels into vertical position for stoping.

THOR POWER TOOL COMPANY

Atlanta	Chicago	Detroit	Newark	St. Louis	Export Division,
Birmingham	Cincinnati	Houston	Long Island City, N.Y.	San Francisco	New York City
Boston	Cleveland	Los Angeles	Philadelphia	Seattle	
Buffalo	Denver	Milwaukee	Pittsburgh	Toronto, Canada	



Ni-Hard liners save labor, cut maintenance and grinding costs by providing the utmost in tonnage life. Results obtained in our own grinding aisle show economies you can count on for your own mills.

Ni-Hard shell liners wear only .018 lb. per ton grinding more than 3 million tons of nickel ore

This high tonnage life demonstrates the outstanding performance of Ni-Hard* nickel-chromium white iron shell liners in the concentrator shown above.

Only recently, two sets of lifter bar liners were removed from rod mill service in this plant, Inco's Creighton concentrator, which began operating in 1951.

Used in large mill

The Ni-Hard shell liners served in 10′8″ x 13′ mills grinding highly abrasive nickel ores with 3″ and 3½″ rods. Mills ran at 60% of critical speed. At the discharge, solids in the slurry ran 70%. Total life in the #1 mill was 3,430,000 tons of ore; in #4 mill, 3,266,000 tons. The liners for each mill originally weighed 60,912 lbs.

This example of Ni-Hard shell liner service is a severe one . . . nickel ores are among the toughest. But the record of Ni-Hard liners in such service shows what this material can do for your grinding operations.

Where your abrasion is severe . . . use Ni-Hard liners.

*Registered Trademark

INCO TRADE MARK **THE INTERNATIONAL NICKEL COMPANY, INC.** 67 Wall Street New York 5, N.Y.

Government Aid for Selenium Exploration

Selenium ore has been added to the list of minerals for which the government will grant financial aid for exploration work. Defense Minerals Exploration Administration will advance up to 75 pct of cost of projects to obtain the ore. Companies receiving advances must repay at the rate of 5 pct of net sales but repayment is required only if the ore is mined.

U. S. Gains Say in West Coast Borax Deposits

Borax Consolidated Ltd., a British-controlled company will turn over West Coast borax mines to Pacific Coast Borax Co., a new American firm. American investors will have 8½ pct interest in new firm, but control will be retained by Borax Consolidated. Borax has increasing strategic importance in rocket developments.

No Extension for Canadian Uranium Plan

Canada's policy for the purchase of uranium ore will remain unchanged since reserves are expected to exceed domestic and export requirements for years to come according to Trade Minister Howe. Expiration date of purchase plan is March 31, 1962.

East and West Build Uranium Plants

The first commercial primary uranium processing plant in the eastern U. S. is to be built as an addition to the Canonsburg, Pa. plant of Vitro Rare Metals Co., a division of Vitro Corp. of America. Facilities for a recently developed solvent extraction process will be included. . . . Union Carbide & Carbon Corp. has announced plans for construction of a new uranium mill at Rifle, Colo. and for two ore-receiving stations and chemical upgrading plants, one near Slick, Colo. and another at Green River, Utah.

New Proposal for Geophysical Locations

A new bill which would allow claimstaking as a result of geophysical work in addition to the discovery of minerals in place has been recently introduced. Such bills have been introduced in the past without creating enthusiasm but the increasing interest in geophysical prospecting evoked by the expansion of the uranium industry may cause a change in sympathy.

Calumet & Hecla Forms Canadian Subsidiary

Calumet & Hecla of Canada Ltd. is a new wholly-owned subsidiary of Calumet & Hecla Inc. and will represent an investment of more than $7.25 million. A mill for nonferrous tubular products is scheduled for completion prior to Dec. 31, 1957.

CF&I ROCK BOLTS

cut temporary drift support costs one-third

Climax Molybdenum Company is using CF&I Rock Bolts and Realock® Chain-Link Fabric to save 30-35% on temporary slusher drift support. Developed by the Climax Planning and Mining Departments, the new method:

- Enables miners driving drifts to do their own rock bolting *with the same machines used to drive heading.*
- Is much faster and more versatile than older methods.
- Has been fully tested for safety.
- Affords proved savings over old methods (35% less than 8″ x 8″ square sets; 30% less than wedge-type bolts and landing mats).

The CF&I Rock Bolts used by Climax are 7 ft. long ¾ inch diameter bolts with Pattin Expansion Shells. They have left-hand threads to allow tightening by the left-hand rotation of the drilling machines. A special adapter consisting of a machine chuck and square socket welded together is the only special accessory needed for this operation.

This is just one of the many jobs being performed by CF&I Rock Bolts—known throughout the mining industry for their dependable performance. Why not get the full details on how you can use CF&I Rock Bolts to advantage in your own mining operations? Just write the nearest sales office listed below.

EXPANSION SHELL TYPE ROCK BOLT WITH PATTIN SHELL

Here CF&I Rock Bolts and Realock Chain-Link Fabric are being installed for temporary drift support.

CF&I

Other CF&I Steel Products for the Mining Industry

Cel-Wic Industrial Screens • Wickwire Rope • Grinding Rods • Grinding Balls
Light Rails and Accessories • Realock Chain-Link Fabric

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Quality Gardner-Denver cost-cutters for development and production

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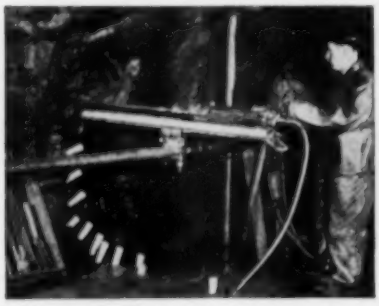
De-Watering, Grout and Sump Pumps



Deep Hole Drilling Equipment

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Fast, Safe Mine Car Loaders

See your Gardner-Denver mining equipment specialist — or write for bulletins.

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GISMO METHOD SETS NEW

New Production Records and Drastic Cuts in Man-hour Costs Foreseen!

COULD THIS BE THE BEGINNING of the "Gismo Age" in rock mining? One mine operator has said: "The Gismo is going to revolutionize rock excavation methods in much the same manner that large tractor drawn earth moving scrapers have revolutionized the earth moving procedure."

One new mine now being mechanized by the Gismo Method contemplates a production of 2,000 tons per day at an overall cost of about $1.00 per ton. Production is forecasted to be approximately 50 tons per man when operated on a 2,000 ton per day basis and 57 tons per man on a 4,000 ton basis. The deviser includes all men to be employed for production, surface and underground, including all supervisors. The above projection is based on results obtained over a three year period by this system in a similar mine.

An older mine, changing over to the Gismo Method, has increased tonnage per man-shift almost 10 times in the stoping operations. Compared to the conventional method used in 1950, the Gismo Method has halved both development and labor costs.

What is this equipment? It is a general purpose utility self-loading transport that is simple, versatile and built to take the punishment of rock excavation. No special development facilities or conditions are usually required. It operates in sloping ore bodies with irregular outlines, as well as large or relatively small openings. It loads in development or production . . . transports . . . supports 2 to 5 jib mounted drills . . . back fills, moves boulder rocks . . . makes its own roadways and cleans up completely. It can load and transport up to 100 tons per hour with a 300 ft. haul.

It's an extremely simple piece of equipment requiring little maintenance. The Gismo's initial cost can be reasonably compared with one year's maintenance cost alone of much of the conventional equipment in present use.

It makes possible a mining method requiring few machines, few men, less planning and integration. It permits total mechanization of your mine—the solution to high man-hour and materials cost.

We urge you to compare the Gismo Method with other methods—from initial investment to overall effect on your operation's efficiency. Call, wire or write us today! *Sanford-Day Iron Works, Inc., P. O. Box 1511 . . . Telephone 3-4191, Knoxville, Tennessee.*

Sanford-Day IRON WORKS
KNOXVILLE • TENNESSEE

AND BROWN-FAYRO DIVISION

MINE CARS, All Types - PRECISION WHEELS - "Brownie" HOISTS
CAR RETARDERS - SPOTTERS
PUMPS - OIL SPRAY SYSTEMS

LOADING (MUCKING)— a one-man operation! The speed of the GISMO Self-Loading Transport can be fully appreciated and understood only when seen in actual operation. With sufficient ore and ample surge capacity, one GISMO as a self-loading transport and one GISMO equipped as a stope drilling jumbo will excavate out of a hard rock face over 400 tons in a six hour working time shift. The GISMO handles boulders . . . sets them aside for secondary blasting . . . and does not require a bulldozer, road grader or any other accessory device. The operator has complete control of his shoveling grade—can always look directly at the dipper edge. The shoveling action is gentle. Carrying capacity of the GISMO shown working above is 6 tons. Other capacities are available.

GOALS FOR ROCK MINING!

DUMPING: direct into surge pocket . . . skip pocket . . . conveyor for truck loading . . . or mine cars with ramps (for development work). GISMO above is dumping load into surge pocket. Both when mucking (loading material into body of GISMO) and when dumping the load, the dipper or shovel head is in position shown above. The surge pocket disconnects from a time cycle standpoint the next operation, transportation—permitting GISMO and the transportation system to operate independently of each other for maximum efficiency and production.

TRAMMING: fast and highly maneuverable! GISMO shown above is transporting load to a surge pocket. When emptied, GISMO will return to reload with dipper in position shown. As you can guess, the GISMO is a very simple machine functionally and mechanically. Compared to any other loading or mucking device (none of which are transports), the GISMO has a very large capacity.

DRILLING: The GISMO will mount 2, 3, 4 or 5 drills at ideal locations for the efficient drilling of small or large headings, benches, rib-slabbing or back-slabbing. The GISMO Stope Drilling Jumbo mounting four drills and operated by two men often averages by the month over 400 feet per man shift. The Gismo Drill Jumbo can be easily converted to a Gismo Self-Loading Transport.

STANDARD OF COMPARISON AROUND THE WORLD
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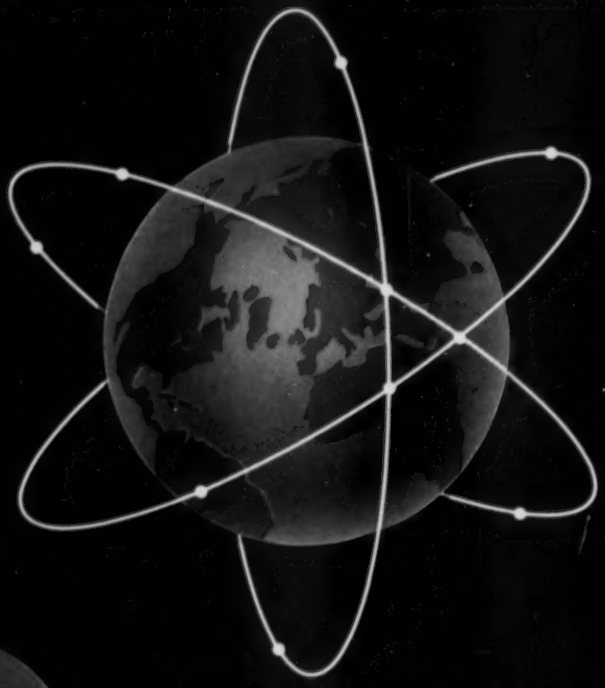

FOR OVER 20 YEARS!

SHEFFIELD

ARMCO

MOLY-COP

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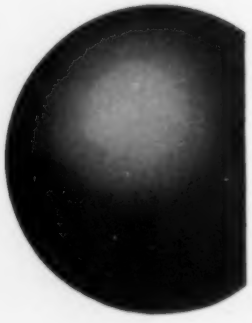
Grinding Balls



SHEFFIELD STEEL

DIVISION

ARMCO STEEL CORPORATION

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Heads Together

At Allis-Chalmers, Integrated Thinking Precedes the Application of Your Integrated Equipment

Allis-Chalmers designs and builds diversified yet completely integrated lines of processing machinery as well as electrical generation, distribution and utilization equipment.

Take a look at the various product departments at Allis-Chalmers and you'll find engineering offices set up for *each* product line. Peek into conference rooms and chances are you will see members of these specialized staffs huddled on application problems . . . exchanging ideas . . . correlating specific know-how and skills.

This proven approach — for which Allis-Chalmers has long been famous — results in truly integrated equipment . . . installations with the various components matched for maximum efficiency at low cost. A-4728

MORE ABOUT ALLIS-CHALMERS INTEGRATION ON FOLLOWING PAGES.

ALLIS-CHALMERS

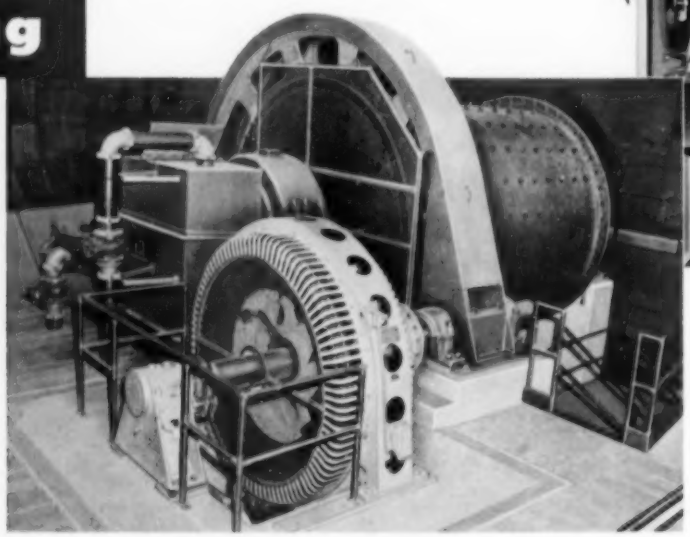
Milwaukee 1, Wisconsin

Allis-Chalmers *Teamed Up* for Grinding

BEFORE A RECOMMENDATION IS MADE, requirements and variables are given a careful going over by an Allis-Chalmers team comprised of *grinding*, *motor* and *control* engineers. Characteristics of material, capacity, feed preparation, balance of gradations, torque characteristics, system power factor needs, control requirements are some of the many factors evaluated. Experience has proved that this thorough pre-application investigation and preparation pays off to the purchaser . . . *pays off in maximum capacity of desired product per cubic foot of mill volume.*

A Typical Integrated Grinding Mill Installation

In a recent application, the mill installed was a 10½ by 12-foot Allis-Chalmers diaphragm ball mill. The close diameter-length ratio is an important factor in producing the highest possible capacity per unit of power. Driving the mill is a 900-hp, 4000-volt, 257-rpm, .8 pf Allis-Chalmers synchronous motor. By providing desired power factor correction, this motor reduces power cost. The Allis-Chalmers starter is specially engineered to provide protection under all conditions of grinding operation.



Modern Electrical Distribution Systems Cut Costs

Allis-Chalmers unit substations can be strategically located to bring high voltages near load centers with resultant economy in cable cost, minimum line loss and efficient voltage regulation. Illustrated is Allis-Chalmers switchgear installed underground where it serves motors driving crushers, compressors, shovels and drills.

At Allis-Chalmers, electrical and mechanical engineers combine their skills and knowledge in coordinating the electrical distribution system with processing equipment when working out an efficient product flow.

This integrated effort results in maximum plant efficiency — reduces installation and power costs.

Equipment
Profitable Production

TODAY, WITH SUCH FACTORS as high kw consumption, frequent lengthy outages and costly maintenance disturbing the profit picture, the advantages of integration deserve prime consideration. For example, let's take the problem of cutting cost in a crushing installation.

Special Control Circuit

Drawing on 75 years' experience in building and applying crushers, Allis-Chalmers control *and* crushing engineers developed a control circuit designed to cut costly downtime for the crusher installation shown below. In this circuit, motor overload protection is provided by *two* sets of thermal overload relays. One set operates at slight overload to sound warning. The second set stops the motor when temperature reaches the danger point.

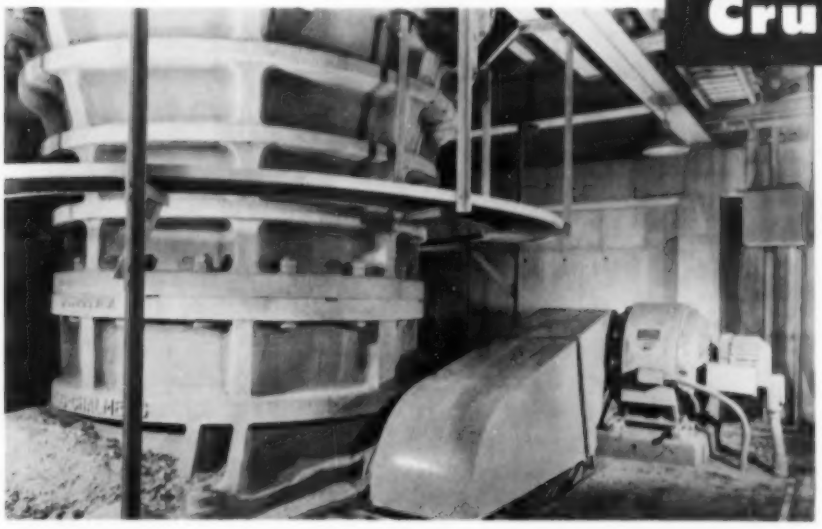
Because of high starting torque and frequent starting under load, an Allis-Chalmers wound-rotor motor is used. In calculating hp requirements, the factors of crushability, ratio of reduction, product size and specific gravity are evaluated.

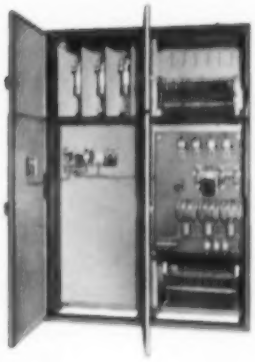
Flexibility in the Crusher

The crusher is the *Superior* gyratory crusher which features hydraulic positioning of mainshaft and mantle. This control facilitates emptying crushing chamber in case of power failure or other emergencies. It also compensates for wear on concaves and mantle and, when required, changes product size instantly. In Allis-Chalmers gyratory crushers, changing eccentricity, speed, or shape of chamber varies capacity and product size. This flexibility permits synchronizing crushing with other operations.

Superior is an Allis-Chalmers trademark.

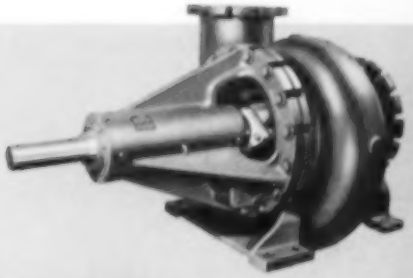
Crushing





Solids Handling Pump Built to Resist Abrasion

Here is a pump specifically designed to meet the problems of mineral processing. Rugged construction featuring the use of abrasion-resisting materials provides longest possible service life. Because some wear is inevitable in handling mineral solids in suspension, inspection and maintenance have been made easy.



ALLIS-CHALMERS

Your *Dollar Buys More* when you invest in Allis-Chalmers Integrated Equipment

Because:

- Integrated units or processes evolve from a co-operative engineering effort — all Allis-Chalmers components are matched for top efficiency.
- New as well as *existing equipment* is integrated into a smooth, productive, profitable operation.
- Allis-Chalmers assures undivided responsibility for performance — there is no "buck passing" as frequently prevails when components are purchased from more than one source.
- No other company can provide the background of experience gained through the years of solving thousands of equipment integration problems . . . in research...in testing...in pilot plant operation.
- Modern laboratory facilities, periodic equipment checkups, and fast emergency service from a large stock are always available.

Cost is more than a matter of price tag. Be sure you are getting all the above advantages.

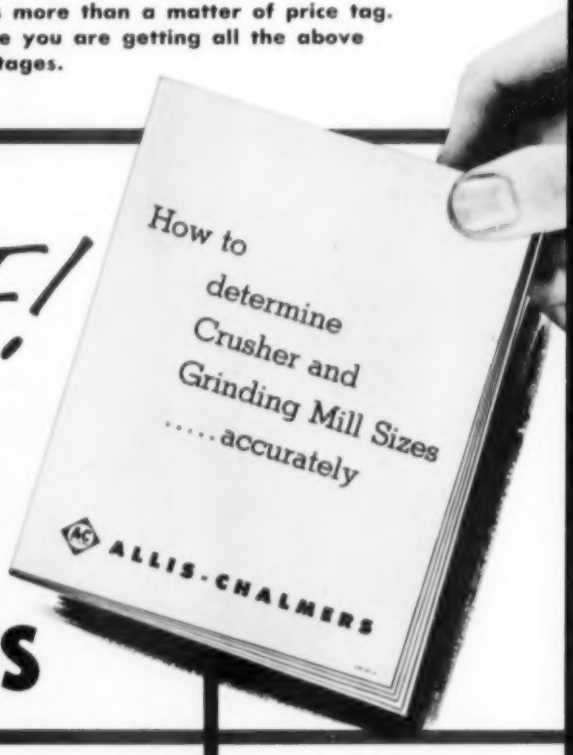
Only from Allis-Chalmers

. . . and it's *FREE!*

Nowhere else can you get this valuable book at any price!

The Third Theory of Comminution, or *work index* method, is a compilation of factual data on crushing and grinding obtained in over 35 years of field and laboratory testing. Correlating this information made it possible to establish a consistent common factor called the *work index*. Knowing the *work index*, you need only apply a given equation to determine required power input. The calculated power input enables you to select the proper crusher or mill for your job.

This practical formula, together with more than 1200 tabulated work indexes, is now available in a 72-page bulletin (07R7995). For your cost-free copy, see your Allis-Chalmers representative or write Allis-Chalmers, Industrial Equipment Division, Milwaukee 1, Wisconsin.



ALLIS-CHALMERS

INDUSTRIAL EQUIPMENT DIVISION

ALLIS-CHALMERS Milwaukee 1, Wisconsin

Send me the following bulletins:

- 25C6166D, a mining industries catalog
- 07R7995, Third Theory of Comminution

I would also like bulletins on:

_____

Explanation of Process or Problem _____

_____
(Type of Equipment)

Name _____ Title _____

Company _____ Address _____

City _____ Zone _____ State _____ A-4728

WRITE FOR LITERATURE

Bulletin 25C6166D covers all equipment manufactured by Allis-Chalmers for the mining industries. Individual bulletins covering specific equipment lines are also available.

MINE VENTILATION

MINING engineering NEWS

Stobie Mine Uses Icing System to Warm and Cool Air

A unique air conditioning system that raises the temperature of cold air in the winter by making ice and uses the ice to reduce summer heat has been put into service at the International Nickel Co.'s Frood-Stobie Mine in Ontario. Basis of the unusual system is the convenient twist of nature whereby heat is released when ice is formed.

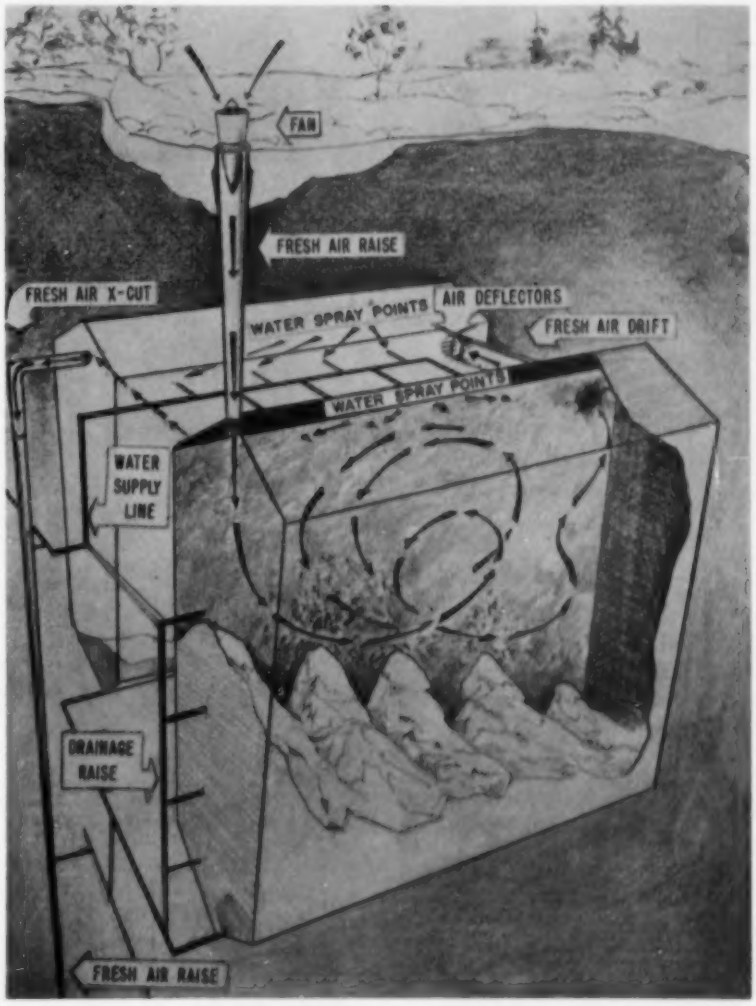
A fan blows fresh air from the surface through a 300-ft vertical passage of 20-ft diam into two huge open stopes between the 300 and 500-ft levels, and through to the main intake. Water is furnished to four spray points at the top of each stope—the volume in winter varying up to 200 gpm. The sprayed water turns to ice in the frigid air and in the process gives up heat. Some 40,000 tons of ice were formed at the bottom of the stopes, during the past winter. Heat, along with that absorbed by the air from the large area of wall rock exposed in the stopes, added up to 15 billion Btu—the equivalent of burning 100,000 gal of oil.

Although surface temperature ranged to −25°F, the fresh air, forced 600 ft below surface, varied only between 27 and 30°F. As system is improved, a yield bringing temperature to 32°F is expected.

When the volume of air handled through the system ultimately reaches full capacity of 750,000 cfm, upwards of 140,000 tons of ice will be formed in the stopes during each winter.

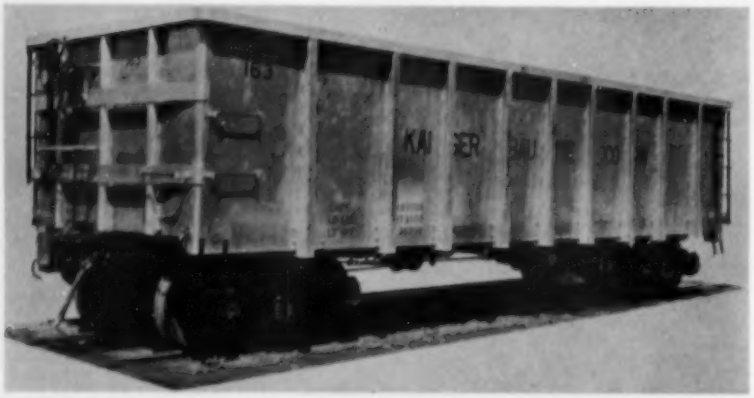
In the summer the ice will be melted as the warm fresh air passes through the stopes, and the air will be cooled between five and ten degrees.

System was established at a fraction of cost of a surface heating plant.



Surface fan forces fresh air over cavernous stopes to main intake. Ice is made in winter by spraying water at top of stopes, and latent heat of freezing is transferred to air. In summer warm air is cooled over ice and water is drained to the pumping system. Stopes are 80 ft wide, 200 ft long, and 200 ft high.

One of the world's largest, the 198-in. diam fan at right uses a 650-hp motor to deliver fresh air to Frood-Stobie Mine. Impeller shaft spins 12 adjustable blades that can handle up to 750,000 cfm.



Aluminum Gondola Cars

All-welded aluminum gondola car at left is one of 35 such cars built for Kaiser Aluminum & Chemical Corp.'s bauxite mining operations on the Island of Jamaica. First of their kind built in the U. S., the cars will have a payload increase of 14 tons per car over conventional steel gondola cars now used by the company.

In their 13-mile-long run from the bauxite mines to Port Kaiser on Jamaica, the aluminum cars will withstand the corrosive effect of a marine atmosphere, the abrasive action of bauxite, and the severe stresses imposed by rotary dumping.

Contact areas where the aluminum body joins the 70-ton steel truck are protected by the use of zinc chromate paint on the aluminum and a red lead application on the steel. Entire underframe is also sprayed with a car cement.

Four large 7/16-in. aluminum plates, butt-welded, form the car floor. Drain holes of ¾-in. diam are incorporated in the floor plates to prevent the collection and retention of moisture within the car body. Side sheets are sloped outward at a 2° angle to facilitate gravity dumping.

CONTRACT CORE DRILLING

EXPLORATION FOR MINERAL DEPOSITS
INCLUDING URANIUM & LIMESTONE — ANYWHERE

FOUNDATION TEST BORING • GROUT HOLE DRILLING

Skilled crews and complete stock of core drills and accessory equipment maintained at all times

Core Drill Contractors for more than 60 years

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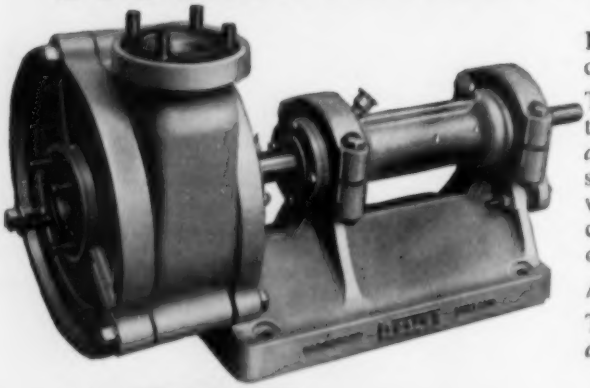
"Acettaremos una Gruesa"



In the Argentine it means "we'll take a gross". In this case a gross of Linatex Sand Pumps.

The Argentine needed a large number of pumps to meet the growing demands of their Mining Industry. Not just *any* pumps, but installations that could handle abrasive slurries without wearing out; that could pump sand and water with a maximum of 20% solids by volume at 25 cubic metres per hour against a total manometric head of 24 metres.

A gross of *Linatex* pumps were ordered.

That's a big order—and a sound vote of confidence, in *any* language.

The LINATEX Pump—*the most lasting pump in the world*

AUSTRALIA Linatex (Australia) Pty. Ltd., David Street, Dandenong, Victoria, Australia	6 of the 15 Linatex factories in the world Any of them will see that your enquiries receive energetic action		**U.S.A.** Linatex Corporation of America, Vernon Avenue, Rockville, Conn. U.S.A.
CANADA Wilkinson Linatex Co. Ltd., 2210 Parthenais St., Montreal 24, Quebec, Canada	**ENGLAND** Wilkinson Rubber Linatex Ltd., Camberley, Surrey England	**MALAYA** The Wilkinson Process Rubber Co. Ltd., Batu Caves, Selangor, Fed. of Malaya	**STH. AFRICA** R. J. Spargo Ltd., P.O. Box 7128, Johannesburg S. Africa

P.18(M)

Allis-Chalmers

model D

THE LOW-COST MOTOR GRADER THAT OFFERS

more of everything...

more PRODUCTION-BOOSTING FEATURES

Choice of Allis-Chalmers engines

GASOLINE — 50 brake hp. Bare grader weight —8,800 lb. Four forward speeds to 25.6 mph, one reverse to 3.3 mph.

DIESEL — 50 brake hp. Bare grader weight — 9,350 lb. Four forward speeds to 25.2 mph, one reverse to 3.2 mph.

ROLL-AWAY moldboard rolls the load instead of pushing it, moves more dirt, uses less power.

Plus one-piece tubular frame, tandem drive, real operator comfort, hydraulic blade lift.

USEFUL OPTIONAL FEATURES

Power circle turn for faster, easier, better control of blade . . . turns 135 degrees.

Leaning front wheels with heavy-duty axle, for better control in close quarter-work.

Hydraulically shiftable moldboard to speed work around obstructions. Shifts 16 in. each way.

Special large tires for added traction, long wear. Front (7.50-20 or 8.25-20) and rear (8.25-20).

more JOB-MULTIPLYING ATTACHMENTS

Hydraulic scarifier, mounted amidships for maximum effective ground pressure.

Shoulder maintainer for safe, one-pass operation. Also available: one-pass windrow eliminator.

Rear-mounted 5/8-yd loader for fast, low-cost handling of materials.

plus snowplows, completely enclosed standup cab, hydraulically operated mower.

more SATISFIED USERS

Since its introduction in 1949, thousands of satisfied owners have *proved* the usefulness and low-cost versatility of the Model D on all kinds of construction and maintenance.

Call your Allis-Chalmers Construction Machinery Dealer for a demonstration today!

ALLIS-CHALMERS, CONSTRUCTION MACHINERY DIVISION
MILWAUKEE 1, WISCONSIN

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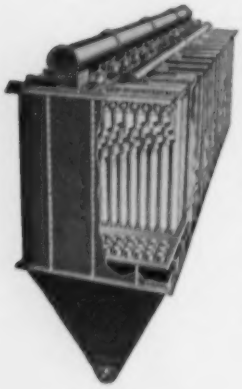
ROLL-AWAY is an Allis-Chalmers trademark.

Norblo solves dust problems

Whatever you need in dust or fume control, be sure to check *Norblo*

In Bag Type, Centrifugal or Hydraulic dust collecting systems, Norblo Dust and Fume collection equipment has many design advantages. These three types, frequently used in certain combinations, provide economical dependable control as required in smelting, rock products, chemical, milling and processing fields.

What is the best approach to your dust problems? Norblo helps you select—applies engineering skill and "know how" gained in first hand experience of over 40 years in many industries.



Norblo Bag Type Systems

Automatic and Standard

For continuous or heavy duty service providing very high efficiency at very low cost of operation and maintenance. Basic unit contains 78 bags, 6" diameter, 8' 3" long. Air flow is upward, from inside, thus keeping bags fully distended. Total free cloth area per compartment 936 square feet. Shaking and cleaning controlled by electric timer, is cyclic, one compartment at a time, each having its individual compressed air shaker mechanism and the whole system variable and adjustable for dust load without shutting down . . . Also Norblo Standard Bag Type (non-automatic).

Norblo H. E. L. S. Centrifugal Systems

A cyclone or centrifugal type collector for all materials, from sawdust to fly ash; characterized by high efficiency of collection with low static drop. The Norblo H. E. L. S. has no internal vanes, gadgets or dampers. High efficiency is obtained by scientific proportioning and by the patented (No. 2,259,919) expanding nozzle. These design features eliminate the power-wasting back eddy. Built in standard sizes with capacity up to 37,500 cfm.

Norblo Hydraulic Systems

A high efficiency, wet type collector, for separation of dust mixed with smoke or fumes. In most installations the Hydraulic unit is used with a Norblo Cyclone collector, thus reducing the amount of wet sludge to be handled. No moving parts. Filter beds are coke or high-fired ceramic tubes, light weight and kept in motion by ascending air stream. Beds are self-cleaning. Built in standard sizes with capacities up to 26,000 cfm.

Take advantage of the efficiency and low maintenance available through Norblo Equipment in these three proved systems of dust and fume collection. Write for full information.

The Northern Blower Company

6424 Barberton Ave., Cleveland 2, Ohio • OLympic 1-1300

Norblo ENGINEERED DUST COLLECTION SYSTEMS FOR ALL INDUSTRIES

Diesel Power Underground

Since the first locomotive was used on the Delaware Aqueduct project in 1939 there has been a steady increase in the use of diesel power.

Diesel equipment can be used safely in underground mines provided that certain basic rules are followed. It is necessary to provide adequate natural or mechanical ventilation, use the proper fuel, maintain and inspect equipment at regular intervals, and provide gas detecting equipment to insure proper atmosphere for employees.

Generally, flat, horizontal ore bodies that utilize the room and pillar method of mining lend themselves to the use of diesel powered machinery. On the other hand, narrow and steep inclined ore bodies are not nearly as adaptable to this type of power and application of equipment is limited.

The second basic requirement for the use of diesel equipment in underground mines is proper ventilation. A gasoline engine produces as much as 17 pct by volume of carbon monoxide while a diesel produces less than 0.25 pct by volume of the same gas. Other exhaust products can be unpleasant and irritating and a wet-type exhaust conditioner is recommended.

Note: The above remarks are excerpts from a paper by Howard A. Raivio, a special representative of Sales Promotion Dept., Caterpillar Tractor Co.

UPPER: Cat No. 6 Traxcavator loads gypsum into an Athey side-dump wagon pulled by a Cat wheel-type tractor at a mine near Shoals, Ind. Current production at mine is approximately 1200 tons per 8-hr shift.

LOWER: Another application of diesel power underground is this Cat D 315 used in pulling a load of gypsum in a mine near Sun City, Kansas.

more yards per load

more loads per hour

Gets More

A combination of powerful "pry-out" action using breakout pads as a fulcrum for leverage and a 40° bucket tip-back *at ground level* gets BIGGER LOADS with less spillage.

Keeps More

Heaped loads are cradled closer and lower for greater stability while carrying. Exclusive load shock-absorber also cushions load, smooths the ride, and permits faster movement with less spillage.

Delivers More

Since you get MORE to begin with and keep MORE while traveling at higher speeds . . . with less spillage in both instances . . . the result—you *deliver* more yards per load and more loads per hour.

Whether you want to move fine manganese ore like this, or larger-size ore and rock, "PAYLOADER" tractor-shovels are designed to get and hold bigger loads and deliver them faster.

"PAYLOADER" superiority on materials handling at mine, mill and smelter is the result of 34 years of pioneering and leadership in tractor-shovel manufacture. "PAYLOADER" preference in the mining industry also comes from the fact that "PAYLOADER" is a complete proven line — a size and type to best meet each need—PLUS the finest parts and service facilities at the nearby "PAYLOADER" Distributor.

PAYLOADER®

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THE FRANK G. HOUGH CO. LIBERTYVILLE, ILL.

SUBSIDIARY—INTERNATIONAL HARVESTER COMPANY

THE FRANK G. HOUGH CO.
916 Sunnyside Ave., Libertyville, Ill.

Send data on "PAYLOADER" Tractor-shovels

- ☐ 4 wheel drive types to 2½ cu. yd.
- ☐ rear wheel drive types to 1½ cu. yd.
- ☐ front wheel drive types to 1 cu. yd.

Name

Title

Company

Street

City State

BOLT a butter-bar-sized chunk of delicate ceramic material into the cutting tool holder of a high-speed lathe. Start up the machine and, if vibrations do not shatter the fragile cutting tool like a teacup, you will be able to make fine cuts to close tolerances in hardened steels at surface-feet-per-minute speeds that would astonish an oldtime machinist. Recently introduced ceramic cutting materials can surpass the speeds of premium cemented tungsten carbides while working modern materials that would quickly dull the edge of these tools. Since they contain no metal binding, the ceramics will not fuse to the material worked even at highest speeds. Details of their composition are being held by manufacturers but aluminum oxide seems to be the common base. Ceramics have the disadvantage of easy breakage—not altogether cured by the use of powerful precision lathes and holding vibration to a minimum. Because they are also expensive (about three times as costly as the finest carbides), the ceramics so far are regarded as just a step in the search for better cutting material, not as a replacement for present tools. But the day when automatic programming controls are commonly used on lathes may be hastened because ceramics can provide the speeds to make use of more complex machines economical.

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TWELFTH annual scholarship award recipients were recently announced by the Princess Elkhorn Coal Co. of David, Ky. The company's grants to outstanding graduating seniors of local county high schools now total 50; and 13 students are currently attending college under the program.

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REDUCTION of the costs of atomic fuels appears to be necessary to the development of atomic energy for power in Canada. In an address at the Seventieth Annual Meeting of The Engineering Institute of Canada, I. N. MacKay, manager of engineering, Civilian Atomic Power Dept., Canadian General Electric Co. Ltd., stated that Canadian reactors must be built to use natural uranium unless imported fuel is relied on. Development of central station atomic power plants seems to be the trend in Canada and economic reasons make the building of expensive enrichment plants unlikely unless part of the cost can be charged to military expense.

"Fumed uranium oxide is currently being studied as a substitute for uranium metal. In large quantities it might be expected to cost less than the pure metal," said Mr. MacKay. He indicated also that the search for substitutes for high-cost reactor materials will be helpful. "For example, an alloy of aluminum and nickel is being actually developed in place of the zircoloy now used for sheathing uranium, which if found suitable would certainly reduce the cost of sheathing the fuel very greatly," he said.

"Thorium as a fuel is another intriguing possibility. It has properties corresponding to uranium 238 and can itself support a chain reaction. Theoretically a reactor fueled with thorium and a small amount of uranium 233 should make possible very long irradiations, perhaps ten times those from natural uranium," he continued. "This fuel might cost less than a cent per kilowatt hour. There are large quantities of thorium in Canadian uranium deposits but extraction might be more costly than obtaining it from monasite sands found in Brazil and India."

Adaptability of the present program was noted by Mr. MacKay who commented that adjustment would be possible in the event of the construction of a diffusion plant to enrich the fuels.

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NEW figures for subdividing the time of man's presence on the earth have been found by scientists of the U. S. Geological Survey through the use of a modification of the *atomic clock* method.

They learned that Late Magdelenian Man for example was not quite as smart as those who like to claim him for an ancestor believe him to have been. In spite of his beautiful colored paintings on the walls and ceilings of certain caves in Southern France, a carbon check gives evidence that he was still a cave man as late as 10,000 years ago. In another test, woven rope sandals found in volcanic ash in an Oregon cave showed their owners to have lived there 9000 years ago, a much earlier date for human life in North America than hitherto accepted.

Atomic clocking is based on the rate of decay of radioactive carbon 14. This is constantly manufactured from atmospheric nitrogen through cosmic ray bombardment, and is assimilated as carbon dioxide by living things. The percentage of these *tagged atoms* is known and can be used to approximate how long ago a plant or animal lived. Two USGS physical chemists perfected a method of preventing air molecule contamination by converting their samples to carbon dioxide and then to acetylene which they freeze out at liquid nitrogen temperature and finally collect as a gas. When determinations are made, cosmic ray background is eliminated by counters that cancel out foreign radiations.

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LARGE cutbacks in automobile production this year have caused a corresponding drop in bids

for special high-grade zinc, points out Clinton H. Crane, chairman of the board, St. Joseph Lead Co. At the annual meeting of stockholders he said surplus metal would tend to be taken up by the government stockpiling program and prices will remain at present level.

He also pointed to U. S. lead consumption in 1955 as the second highest in history, reaching a level of 1.2 million tons. Mr. Crane felt sure that there would be no shortage of lead during the remainder of 1956 and commented that U. S. smelter production of lead for the first quarter of 1956 is higher than the first quarter of last year.

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BRITAIN'S coal problem is probably the darkest blot on an economy that has tended to increase in strength in recent years.

Energy is a major concern of modern economy, and, despite the advances made toward the development of new energy sources, coal remains the cradle of industrial power. The nationalized British coal industry went still deeper into debt last year even though its collieries operated at a profit. According to the annual report of the National Coal Board the loss was incurred in the sale of imported coal. The accumulated deficit was almost doubled because of the loss of an average of $6.60 a ton on these imports in 1955. The NCB has said that capital expenditure of £1 million in the next ten years on equipment and new mines would be necessary because of the decline in production in mines now operating.

Behind the general problem is the fact that coal is an industry that requires increasing capital investment to maintain rather than increase its productive capacity. For this reason, one strong suggestion has been to increase fuel efficiency. The claim is that time and money required to open more and more new collieries would in part be saved.

Mechanization is also seen lacking. While the percentage of coal cut by machine has increased in Britain, shot-firing and loading in normal practice continue to be manual operations. Investment in the development of new equipment is claimed to have brought results, however, and the NCB is now said to be in a position to apply power-loading on a wider scale.

Iron & Coal Trades Review of London sees another problem. "A lot of money has been spent—£323,000,000 since 1950—and although much of it is still 'in the pipeline,' it cannot be claimed that the mines have been starved of equipment. . . . Output is plainly not what it should be, and in line with technical development there must go a drive to improve morale and instil a new spirit, a new urge to recreate the industry in everybody connected with it. Internal relations are the weak spot and until they are improved the industry will not get the best out of expenditure on technical re-equipment."

The need for increased investment remains a certainty; output would inevitably drop without it and results are awaited from the money spent so far. Rate of British coal production this year is at approximately the same level as this time last year but output is greater because less time has been lost through labor disputes. It is expected that results of the effort to offset the natural loss of mining capacity will only appear slowly. If they are coupled with a solution to the need for increased manpower the result may carry British industry to the point when new energy sources can be utilized.

But future roles to be taken by oil and atomic energy were estimated small in the NCB annual report. "In twenty years time, atomic energy may be contributing the equivalent of 40 million tons of coal a year. On the assumption of a 2 per cent annual increase in demand, this would leave the equivalent of about 310 million tons to be supplied by coal and oil.

"Much the greater part of the 310 million tons must be provided by coal, perhaps of the order of 80 per cent," it said.

Minister of Fuel and Power Aubrey Jones warned recently, "Do not let us delude ourselves that we can substitute nuclear energy for coal in a few short years. To aim at a coal saving of 50 million tons from nuclear energy over the next 10 years would mean converting every power station now being built, as well as many modern power stations now operating, to nuclear energy—all this before we have had any experience of operating even the rudimentary nuclear power station which is all at the moment we could put down. To stretch our few technologists over such a programme would be an impossible task."

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"MINERAL Facts and Problems" is a new 1048-page encyclopedia made available by the U. S. Bureau of Mines and representing the work of over 50 authorities in mineral production, processing, and economics. Former Secretary of the Interior Douglas McKay stated that "here, for the first time, is found the most thorough Federal report on mineral resources ever assembled in a single volume . . . The encyclopedia is of extreme importance to every person concerned with the well-being of our Nation."

At first a series of special statements designed to aid USBM in advising government officials, the reports were seen to have wide value and it was decided to make them available to the public in a single volume. There are 85 fact-filled chapters which, if desired, may be purchased singly at nominal cost. Necessary ordering information can be found in the *Books* section of this issue, page 668.

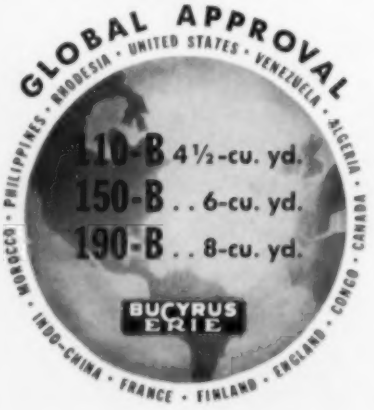
In California...and the World Over
Bucyrus-Eries Live Up to Reputation for High Output

Bucyrus-Erie 6-yd. 150-B shovel working at Eagle Mountain mine. Ore bearing formations here are high in iron content and considerably heavier than most rock.

Miners the world over have learned to expect outstanding performance from Bucyrus-Erie Ward Leonard electric shovels — and they get it, no matter how tough the assignment.

The machine above, for instance, is on a California project where it handles massive, heavy iron ore and waste rock — an extremely hard, fine rock which is quite abrasive. It's a true test of an excavator's strength and durability but here, as on other tough jobs the world over, the Bucyrus-Erie shovel moves big yardages economically — day after day, month after month.

It's the *extra margin of quality,* both in design and manufacture, that qualifies Bucyrus-Erie Ward Leonard electric shovels for world-wide acceptance — and lets them live up to it. We would like the opportunity of explaining that extra margin of quality — and what it can mean in performance on your job. 96L56

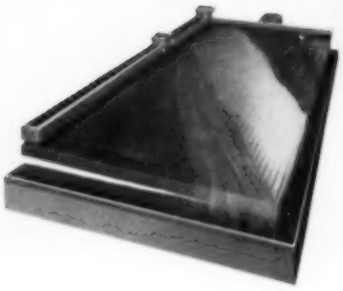


BUCYRUS-ERIE COMPANY

SOUTH MILWAUKEE, WISCONSIN

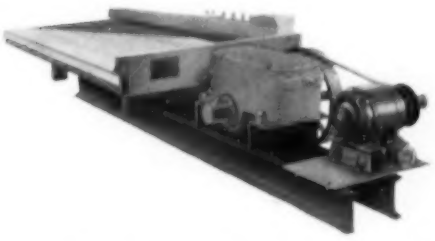
WILFLEY CONCENTRATING TABLE

many profitable applications in modern metallurgy

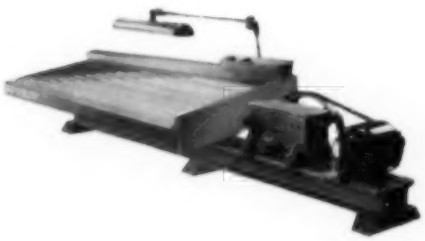


No. 6 Wilfley Table with variable speed V-Belt motor drive.

No. 11-D Wilfley Table showing concrete foundations.



No. 12 Wilfley Table with variable speed V-Belt motor drive.



No. 13 Spec. Wilfley Table with overhead Lamp, as Flotation Pilot Table.

The **genuine WILFLEY TABLE,** as manufactured by Mine & Smelter, has been an important factor in the success of many metallurgical processes for years. There are still many applications where the Wilfley Table is of particular value and cannot be equalled for economical and efficient performance. Some of these are...

- **Before or after flotation for recovery of mineral not readily recovered by flotation.**
- **As pilot tables following flotation to indicate efficiency of flotation.**
- **To concentrate low grade mineral producing a high grade concentrate for further treatment.**
- **To wash and clean inferior coals, iron ore, etc.**
- **In metal and reclamation plants.**
- **For two or more mineral separations from flotation concentrates, previous table concentration or from original table feed.**

The Wilfley Table has an easily adjustable stroke from ⅝" to 1", a speed range from 240 to 300 strokes per minute, an adjustable table inclination from zero to 1" per foot, and a selection of wood riffles of various depth and taper...all of which permit a flexibility to best meet the conditions of each application.

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AERO® Brand Cyanide Available *from Basic Integrated Production*

Cyanamid customers are assured of an unfailing supply of AERO Brand Cyanide to cover their complete needs. Basic, integrated cyanide production in both hemispheres doubly assures continuance of our performance record: "Through two World Wars no Cyanamid customer has ever had to curtail or shut down mill operations for lack of AERO® Brand Cyanide".

Equally important, AERO Brand Cyanide has inherent advantages that make it an excellent money-value:

On the basis of contained NaCN equivalent, the metallurgical efficiency of AERO Brand Cyanide equals that of any other grade.

AERO Brand Cyanide has an inherent protective alkali content which usually reduces lime requirements.

AERO Brand Cyanide is clean, easy to handle and has been used for years with perfect satisfaction by small and large mills alike.

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Our Field Engineers are always available to assist on metallurgical problems. They bring you the combined experience of a mature world-wide technical staff and the research resources of Cyanamid's Mineral Dressing Laboratory.

To further assist AERO Brand Cyanide users we have recently issued

Mineral Dressing Notes No. 22

HANDLING AND FEEDING OF AERO® BRAND CYANIDE

This 16-page manual contains a wealth of practical suggestions and diagrams of feeder arrangements for both small and large mills. A copy will be sent with our compliments.

CYANAMID

AMERICAN CYANAMID COMPANY

MINERAL DRESSING DEPARTMENT

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American Cyanamid Company
30 Rockefeller Plaza
New York 20, N. Y.

Kindly send me_____ copies of Mineral Dressing Notes #22 — HANDLING AND FEEDING OF AERO® BRAND CYANIDE

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Look What's Happened to Classification . . .

Wet classification — important unit operation throughout the process industries — demands special equipment and a wide range of designs. Typical of this trend to specialization is the new, ultra-modern phosphate rock washing plant of Armour Fertilizer Works near Bartow, Florida. No less than nineteen individual Dorr "Classifiers" of five radically different designs are being used for size separation.

1 A Dorr Classifier, upper left, makes a 65 mesh separation with rake product capacity of 240 long tons per hour splitting to two 16½ packet Dorreo Jet Sizers for further classification ahead of screens.

2 Combined overflow from Classifier and Sizers is treated in this 50′ dia. Dorr Type H-1 Hydroseparator.

3 Desliming at 150 mesh is accomplished in fourteen 24″ dia. DorrClone Separators equipped with Vactrol control devices. Units respond automatically to changes in feed . . . produce uniform underflow at 70-75% solids.

Dorr-Oliver is the only manufacturer of a complete line of wet classification equipment. When you call on Dorr-Oliver you can be sure you'll get an unbiased recommendation . . . the *right* equipment to solve your particular classification problem.

4 High capacity Dorr Type AX Hydroseparator handles flotation concentrates prior to further treatment.

DorrClone and Vactrol T.M. Reg. U.S. Pat. Off.

DORR-OLIVER

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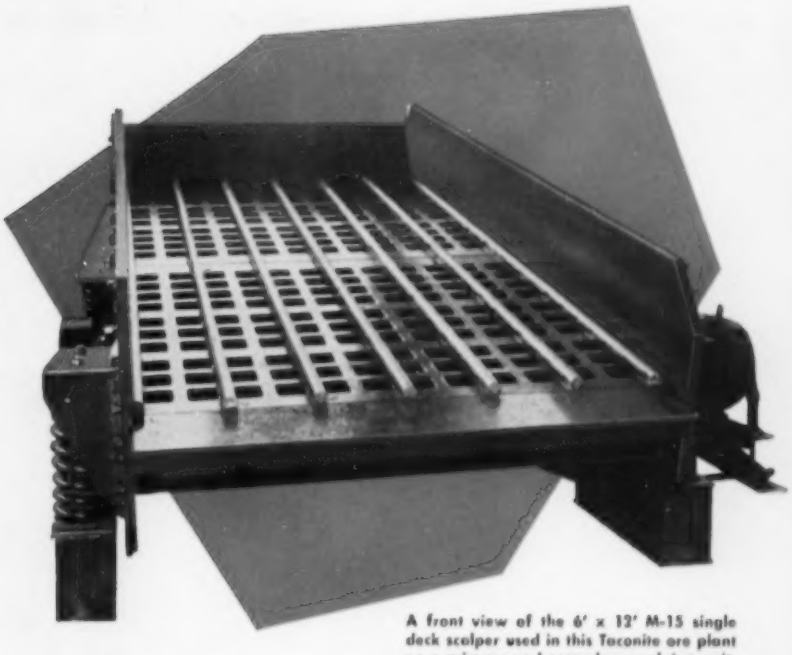
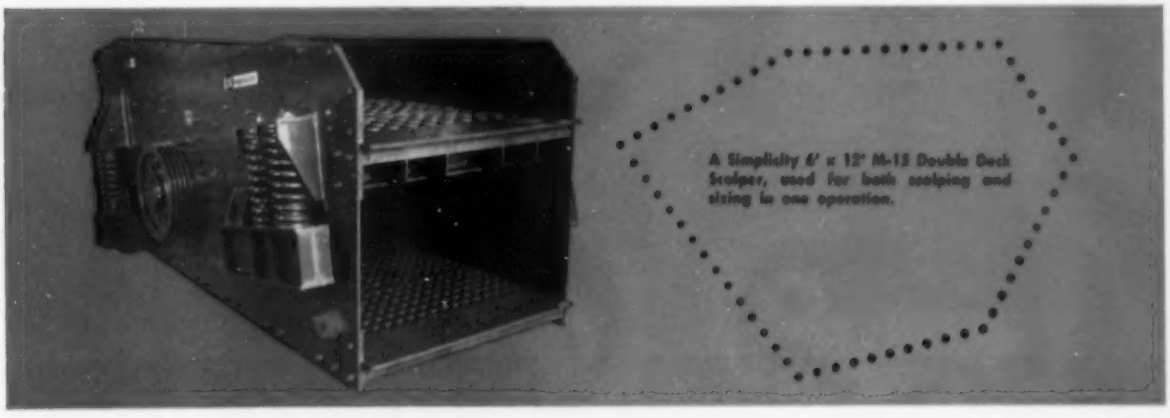
430 long tons per hour of Taconite ore handled by this SIMPLICITY scalper



A front view of the 6' x 12' M-15 single deck scalper used in this Taconite ore plant as a primary and secondary scalping unit.

Lumps up to 24" of hard, tough, extremely abrasive Taconite ore are scalped by this Simplicity 6' x 12' Model M-15, at a throughput rate of 430 long tons an hour. This primary scalper is accompanied on the job by six other heavy duty Simplicity units, all of them, of course, handling the same gruelling fast flow of Taconite. First operation in the Taconite processing is feeding the ore to the primary scalper: this is done by a Simplicity 6' x 16' Model MA Feeder. Large chunks are fed off the 6' x 12' single deck M-15 primary scalper onto a Simplicity 6' x 20' Picking Table, where all obvious shale, stone, and other foreign material is removed. From here, the ore is put through a jaw crusher, and from the crusher two Simplicity 5' x 20' Feeders set in tandem deliver it to the secondary scalper. This unit, like the primary, is a Simplicity M-15 6' x 12' Heavy Duty Scalper. Fines from this operation now go to a Simplicity Double Deck 6' x 12' M-15 Scalper, where they are both scalped again and graded. Larger lumps are fed through a cone crusher, then they are likewise scalped and graded. The ore is now ready for extraction. This whole plant is set to work 24 hours a day seven days a week . . . and the rugged Simplicity equipment assures that down time will be negligible. Write today for details of Simplicity units for your own operation.



A Simplicity 6' x 12' M-15 Double Deck Scalper, used for both scalping and sizing in one operation.

180

SALES REPRESENTATIVES IN ALL PARTS OF THE U.S.A.
FOR CANADA: SIMPLICITY MATERIALS HANDLING LIMITED, GUELPH, ONTARIO
FOR EXPORT: BROWN AND SITES, 50 CHURCH ST., NEW YORK 7, N. Y.

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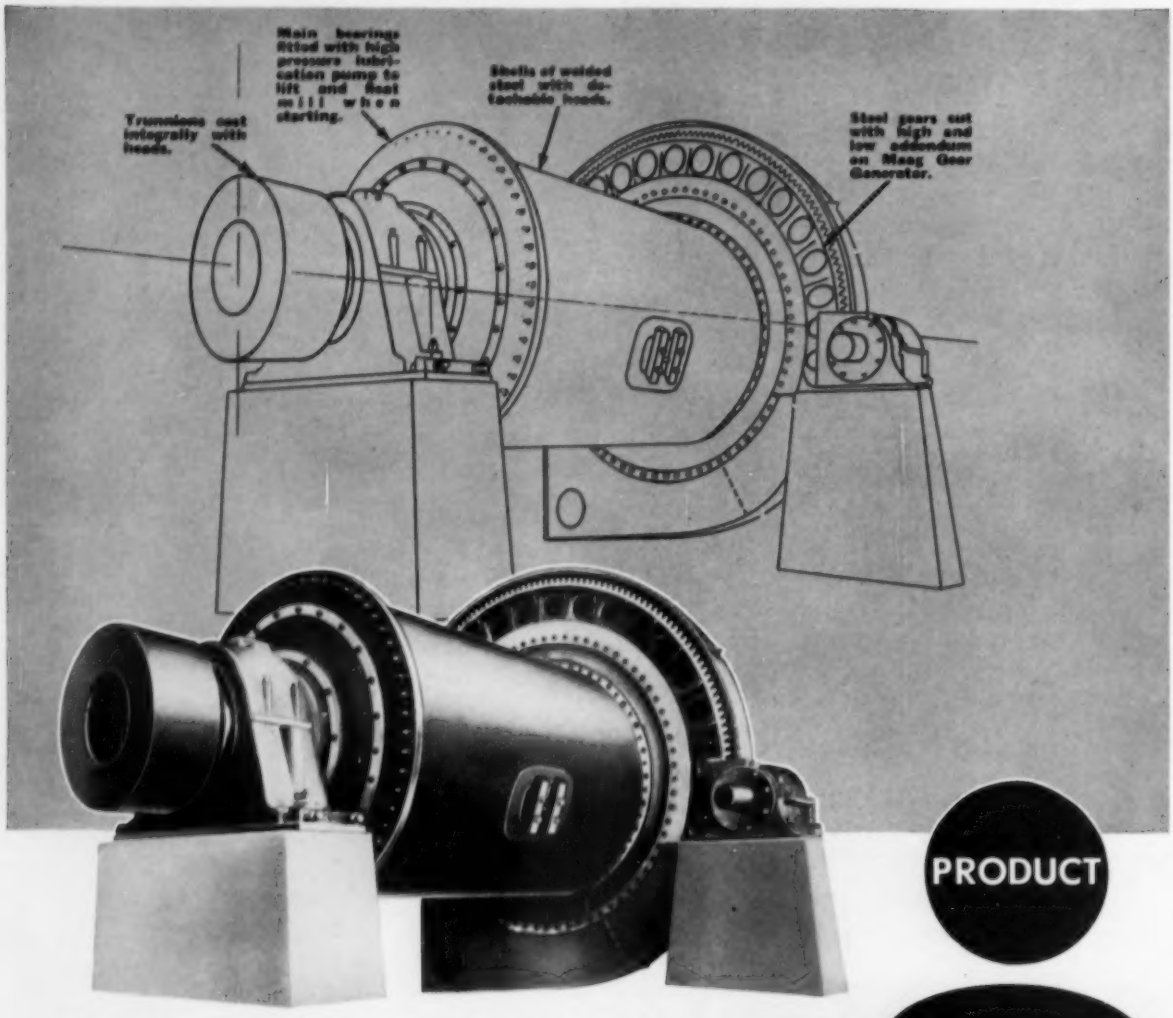
For over a half-century Traylor Grinding Mills have been regarded as the accepted standards of grinding mill design.


Main bearings fitted with high pressure lubrication pump to lift and float mill when starting.
Shells of welded steel with dished heads.
Trunnions cast integrally with heads.
Steel gears cut with high and low addendum on Maag Gear Generator.

Traylor Grinding Mills Improve All 3

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PRODUCTION

PROFITS

Leading producers rely on the many proven features of Traylor Grinding Mills to assure them more efficient production with added production economies.

Available in sizes to meet practically every requirement. Traylor Grinding Mills are offered in a wide variety of types including: Rod, Compartment, Ball, and Tube Mills for wet or dry . . . coarse or fine grinding.

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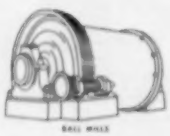
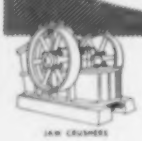
PRIMARY GYRATORY CRUSHERS

ROTARY KILNS

SECONDARY GYRATORY CRUSHERS


BALL MILLS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Outstanding though this production record is, there is now a *new* Caterpillar DW21 (Series C) and No. 470 Scraper with LOWBOWL design which will move even *more* material in *shorter* time at *less* cost per yard. Its new four-cycle diesel engine develops 300 HP (maximum output) and has a Turbocharger which packs in air according to load, not speed, for more working horsepower and greater performance.

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NAME THE DATE... YOUR DEALER WILL DEMONSTRATE

Daniel Cowan Jackling

TO those who had the privilege of working for and with Mr. Jackling—and it was our good fortune to have that privilege for over a quarter of a century—he was affectionately known as "The Boss." He was that—a man among men.

Mr. Jackling was indeed a rare and gifted man. With a vision of a new horizon for copper mining, he possessed the persistent courage, the ability and industry, despite delays, discouragements, and disappointments, to see the vision through and bring it to unbelievable success.

Not the least of the abilities of Mr. Jackling was his inspirational leadership. He himself was a tireless worker, but he possessed the ability to transmit to his co-workers an enthusiasm and devotion to the work at hand so that in heart and soul they were "his team."

Orphaned at the age of two, Mr. Jackling spent his boyhood on Missouri farms with various relatives. He completed his public schooling at sixteen; later, he became attracted to engineering and enrolled at the Missouri School of Mines. A brilliant scholar, he finished the four-year course in three years, graduating in 1892 with a Bachelor of Science degree.

Following graduation, he worked in varying capacities—as miner, mill hand, assayer, metallurgist, and as construction and metallurgical superintendent.

Engaged to examine the copper ore deposit at Bingham Canyon, Utah, he became convinced, after exhaustive testing and planning, that the orebody, despite its low copper content, could be worked profitably. His basic idea, or dream, was that the ore—of a grade far too low to be treated by conventional methods—would become of commercial value if it could be mined and treated by mass means. Accordingly, his revolutionary proposal, made in 1899, was that the waste rock overlying the ore be stripped and the ore thus exposed be mined with steam shovels; that, in place of a 500-ton-per-day concentrator, then regarded as a large concentrator, a 2,000-ton concentrator be constructed. Many experienced mining engineers viewed these plans with skepticism and distrust. Because his ideas were so foreign to the then accepted mining practices, financial backing could not at that time be found for the project.

But the dream of developing a mine at the Bingham Canyon deposit persisted. It was not until 1903, however, that others became persuaded, in no small part due to the character of the man, to back the dream and to form the Utah Copper Co. to develop the property.

After some four years spent in preparing and developing an open pit for mining by steam shovels; building not a 2000-ton concentrator, but one of 6000-ton capacity; building a railway system; and providing other production facilities—production was started in August 1907. Attesting to the rightness of "The Boss's" ideas and the care and thought he expended on the plans to bring this new and untried method of copper mining into being, the operation proved successful from the start.

Mr. Jackling has been called the "Father of the Porphyries." The success which attended the development of the low-grade orebody at Bingham Canyon revolutionized copper mining. As a result of Mr. Jackling's pioneering in making low-grade porphyry ores commercial ores, other major low-grade copper deposits in the United States, as well as abroad, have been successfully opened for commercial production. The billions of pounds of copper since produced from porphyry mines in the U. S. and abroad evidence the tremendous contribution to society made possible by this man's vision and his creative work at Bingham Canyon.

Mr. Jackling received many honors during his long, busy, and useful life. He received from President Woodrow Wilson a Distinguished Service Medal in 1919 for his service in World War I as director of all U. S. Government explosives plants.

Four of the world's highest awards ever accorded an individual engineer were given to him. In 1926, he received the Gold Medal Award of the Mining and Metallurgical Society of America for "distinguished service in the development of low-grade mines." The AIME presented the William Lawrence Saunders Gold Medal for Achievement in Mining to him in 1930. He was awarded the John Fritz Medal of the four Founder Engineering Societies in 1933 for "notable industrial achievement in initiating mass production of copper low grade ores, through application of engineering principles." The Washington Award was presented to him in 1940 in Chicago for "pioneering in large scale mining and treatment of low grade copper ores, releasing vast resources from formerly worthless deposits."

Withal, Mr. Jackling was a sincere and generous leader. In his acceptance speech on the occasion of receiving the Mining and Metallurgical Society of America's Gold Medal in 1926, he said:

". . . . . I would not be true to myself, to my profession, my associates, or my friends, including all to whom I now speak, if I failed to accord to my co-workers the credit they merit at least equally with me for advancement in the mining practice, both technical and commercial, on account of which this honorary emblem is now bestowed. . . . . "

This typifies the man. In spite of his individual attainments, he would accept no honor without according to his co-workers "the credit they merit at least equally with me." His brilliancy in his chosen profession was recognized by all. To those who knew and worked with him, their great admiration for his high attainments was no less than their deep personal affection and respect for him as a man and friend.

An Appreciation by

Louis S. Cates and Erle V. Daveler

Memorial Resolution in Honor of DANIEL COWAN JACKLING

WHEREAS, with the passing of Daniel C. Jackling at his home in Woodside, California, on March 13, 1956, the Institute has lost one of its most loyal members; and

WHEREAS, during his fifty-six years of membership he served the Institute as a Director from 1914 to 1916, 1927 to 1929, and 1938 to 1940, and was President in 1938; and

WHEREAS, he was William Lawrence Saunders Medalist in 1930 for distinguished achievement in mining; John Fritz Medalist in 1933 for notable industrial achievement; and in 1941 was elected to Honorary Membership in the Institute; and

WHEREAS, he showed deep interest in assisting young men to enter our profession and in recognizing their progress and achievements;

THEREFORE, BE IT RESOLVED, That the American Institute of Mining, Metallurgical, and Petroleum Engineers records with deep sorrow the passing of one of America's most distinguished engineers, and one of the Institute's best friends and advocates; and

BE IT FURTHER RESOLVED, That this Resolution be included in the minutes of this meeting and that a copy be sent to Mrs. Jackling.

June 20, 1956 **Board of Directors**

SYMPOSIUM ON GREATER COOPER-

Success Depends on:

Geologists

Mining Engineers

Metallurgists

The history of many lead-zinc-copper operations has amply demonstrated that full and continuous cooperation between all three branches of mining pays off. Unwillingness to utilize information supplied by the others, or ignorance of the meaning of the contributions of other departments may impair the work of the team as a whole —and, clearly, one of the chief duties of management is to see that the efforts of the metallurgist, the mining engineer, and the geologist are utilized to the maximum extent for the common good of the mining enterprise.

Contributions that metallurgists, geologists, and mining engineers should make have varying importance during successive phases of a porphyry copper exploration and development program. To be effective, therefore, efforts toward better cooperation should be properly timed.

It is the responsibility of the geologist and mining engineer to supply the metallurgist with pertinent information on ore reserves and such representative samples for testing as are possible from the exploration and mine development stages.

This series of articles is based on a Symposium presented at the Second Annual Rocky Mountain Minerals Conference, Salt Lake City, October 1955.

ATION FOR METALLURGICAL PLANNING

Metallurgist's View on Milling Base Metal Ores

by F. W. McQuiston, Jr.

TEAMWORK is the key to an efficient organization. Cooperation among mining staff members is an obligation owed to the company. The more cooperation the metallurgist receives from underground, the higher milling yield will be, which in turn makes for a greater mine profit. There is no pay for metal left in mill tailings. The metallurgist is not in a position to be of help to either the geologist or mining engineer, but can receive valuable assistance from both. Therefore, the metallurgist must win underground cooperation.

The mining engineer is usually interested in geology because of its relation to ore control; in milling because that is where the rock goes. However, any overlapping of interest between geologist and metallurgist is quite remote.

Cooperation between the metallurgist and geologist would lie in the field of metallurgy, whereas with the mining engineer it lies in the field of mill operation.

Metallurgical results are as dependable as the samples tested. Therefore, the metallurgist should assume responsibility, together with the geologist and mining engineer, for samples being representative.

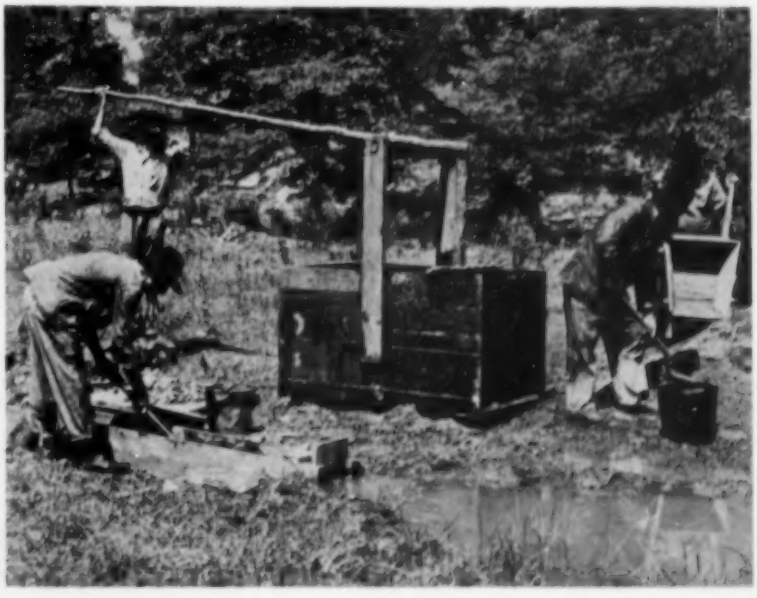
Samples can be divided into two general categories, those for mill design purposes, and those for carefully planned long range metallurgy. Samples for mill design need not be as representative as those used for metallurgical planning. Samples must be representative of future mill feed, as mine earnings will be predicated on test results.

Underground exploration to confirm drilling results and development to block out ore reserves may be sources of representative samples provided horizontal lengths and vertical range are adequate. Samples of this type are usually satisfactory for mill design purposes. A job well done on flowsheet layout, equipment arrangement, design, and construction will give a milling plant capable of yielding good results over a normal range of ore fluctuations. However, to achieve optimum metallurgy over a span of several years requires a suite of samples representing all types of ore and alteration products.

Mine preparation and mill construction usually follow a parallel time schedule. Samples from mine preparation work would be too late to be used in mill design, but can be used in development of metallurgical refinements.

Samples from deposits of simple mineralogical association are not too critical, as mill layout and metallurgical practice will follow standard procedures. However, correct samples from a complex ore such as one containing copper, lead, zinc and iron minerals, with some oxidation, or fine mineral dissemination, are most important.

F. W. McQUISTON, JR. is a Consulting Metallurgist for the Newmont Mining Corp., New York, N. Y.



Modern research laboratories have enabled industry to go far beyond the primitive washing operations using sluice boxes shown at right. But, with specialization and research comes increasing need for interchange of information between specialists—no longer is one man geologist, miner, and metallurgist with all the facts in his head.

Three general categories will probably suffice to define roughly base metal ore deposits from a metallurgical viewpoint: 1) Galena-sphalerite ore with or without oxides of these minerals, but with little or no pyrite. Examples would be deposits in the Tri-State, North Africa, and the lead-silver-zinc ores of the Coeur d'Alene. 2) Fine grained galena-marmatite-pyrite association with or without minor amounts of copper. The Sullivan Mine, Leadville, and the New Brunswick district serve as examples. 3) Galena-sphalerite and copper sulfide minerals with minor to major amounts of oxidation. Examples would be the Colorado San Juans, Tsumeb (Southwest Africa), Cerro de Pasco, and Buchans.

There is more demand for better metallurgical results from a plant milling copper, lead and zinc than from most other types of nonferrous mills. It is rare that this mineral association occurs in deposits of sufficient tonnages for the operation to support three smelters. This means added transportation costs and smelter charges on concentrates sent to custom smelters.

Before the geologist can be of assistance, he must have some understanding of general metallurgical problems. This should include a rough idea of the flowsheet with the metallurgist's interpretation of mineral behavior through the mill circuit. With this background, he should be in a position to develop geological and mineralogical information to aid the metallurgist.

Metallurgical problems revolve mainly around flotation, which is a surface phenomenon, and any alteration on mineral surfaces is important. It should be recognized that metallurgy starts underground. First-stage crushing occurs when the ore is blasted and various phases of underground handling subject the ore to further crushing and, to some extent, grinding. Water percolation and air circulation through broken ore in stopes and ore passes results in uncontrolled *conditioning* of mineral surfaces.

Geological information which would serve in solving metallurgical problems includes the following:

1) **Extent of Oxidation and Alteration:** If oxidation is present, does it persist along the ore trend of strike, and if so, is it apt to diminish with depth?

2) **Products of Oxidation and Alteration:** What are the oxide minerals and alteration products of the sulfides? (Note: Flotation of cerussite, smithsonite and some of the copper oxides is possible).

3) **Rate of Oxidation and Alteration:** Some types of galena are notorious for rapid oxidation or tarnishing. Oxidation of copper minerals is quite critical if zinc minerals are present.

Except for the occasional research-minded metallurgist, the metallurgical engineer often is lost when interpreting microscopic data. Work of this nature should be done by a capable microscopist, under the direction of the geologist, on a mutually agreed program. With a scheme of this kind, the findings will be completely unbiased and will not be viewed with the suspicion of *a metallurgical alibi*.

Microscopic studies can develop a wealth of metallurgical information such as:

1) Identification and occurrence of minerals with estimation of percentages.

2) Extent of mineral dissemination and degree of intergrowth, including estimation of grain sizes and at what micron size liberation should occur.

3) Degree of coating, filming or tarnishing on sulfide mineral surfaces.

4) Occurrences and degree of mineral alteration.

Milling results are often below normal because the metallurgist attempts to make recoveries and grades of concentrates not possible with the ore being milled. A true interpretation of geological and mineralogical data as related to metallurgy would not be an alibi but fact.

Communications:

Mining Engineer ⟶ Metallurgist

• The mining engineer can assist the metallurgist more in the field of mill operation than in metallurgy. For most effective cooperation the metallurgist must become familiar with the planning of the mine program along the following lines:

1) Expected life of the operation.
2) Time schedule for mine preparation and mining.
3) Method of mining and ore transportation.
4) Mining rate.
5) Probable dilution.

Metallurgist ⟶ Mining Engineer

• The metallurgist can develop with the mine engineer information on the following:

Average ore grade and mine cut-off grade.
A daily average grade does not necessarily mean a uniform mill feed. Hourly assays often show variations of more than 200 pct.

Uniformity of feed to mill.
Mixing of different types of ores underground to get an average mine grade can be troublesome. Blending equal portions of two ore types does not always yield average metallurgical results. If fluctuations are wide, some system of blending should be considered.

Characteristics of diluting material.
Dilution is important from both the metallurgical and operating standpoint. When a mining plant is operating at capacity or at a fixed rate, each ton of waste that is broken in a stope and ultimately finds its way to the mill displaces a ton of ore and directly reflects in production figures.

Mining method.
A method involving a large build-up of broken ore reserves, such as from shrinkage stoping, causing a long delay in getting the ore to the mill can have a serious effect on mill results. Best metallurgy is obtained on ore milled with powder smoke in the rock.

Mineral alteration.
As mining progresses and more area is opened, the circulation of air and water can cause sulfide alteration, seriously affecting mill results.

Cooperation in the initial stages of a mining venture is vital for efficient planning. However, success of the operation is dependent upon continued cooperation during the life of the mine.

Cooperation by the geologist and mining engineer should not be taken for granted by the metallurgist. It is the metallurgical department that stands to gain and therefore it becomes the responsibility of the metallurgist to establish proper relations and promote an understanding of and interest in his metallurgical and milling problems.

●●● *symposium*

Cooperation for metallurgical planning

Metallurgical Planning For Porphyry Copper Ores

by J. B. Clemmer

PROFIT in exploiting a porphyry copper deposit is seldom more than a few cents per ton of ore. It is axiomatic that large capital investment is necessary for mining and processing such an ore on a large scale to obtain low production costs. Under such conditions there is no margin for error in technical planning of the operation. The only satisfactory plan must recognize that the objective of the exploitation is to make a maximum overall profit from mining the ore to production of metal.

In nearly every mining district a small abandoned mill of one kind or another is a vivid reminder that mills should not be built before ample resources of acceptable milling ore have been developed. Failure of such ventures usually occurs from lack of funds to do a proper exploration and ore testing job. Success of the porphyry copper operations attests to the fact that the technical staffs have been well aware of the need for concerted planning. This article, therefore, deals primarily with the degree of cooperation necessary to formulate the best plan for insuring a successful commercial operation on a low-grade porphyry copper deposit.

J. B. CLEMMER, Chief, Division of Mineral Technology, Region III, U. S. Bureau of Mines, Salt Lake City.

The metallurgist should be expected to perform a larger part in planning the direction, timing, and scope of the exploration and mine development. The metallurgist seldom is consulted by the geologist during the initial examination of the deposit and may not be brought into the discussion until the initial stages of exploration are well advanced. Too often he first hears about the project after a decision has been made to do detailed exploration and occasionally only after mine development is in progress. The assumption apparently is made that putting ore in the mill bin at the lowest possible cost is the main key to maximum profit. This rarely is so. Both exploration and mine development should take cognizance of such metallurgical information as can be obtained by preliminary as well as final pilot plant study of the ore. Only when all three technologic groups have a voice in all phases of the project will the recommendations made and the plans submitted to management present the technical facts as a properly evaluated and fully integrated proposal.

The Examination Stage: Cooperation and coordination should start when the geologist submits his first samples from examination work for mineralogical study and chemical analysis. In addition to rock classification, mineral identification, and paragenesis studies, typical samples should be examined to determine the nature and complexity of the mineral association of the ore and gangue minerals. If sufficient sample is available, a preliminary flotation test should be made and the concentrate, middling, and tailing products chemically analyzed and examined with a high-power microscope by the geologist or a trained ore-dressing microscopist. The results of such preliminary metallurgical and microscopic studies will aid the metallurgist in planning future research by revealing the presence or absence of unusual milling problems and will benefit the geologist by defining the mineral associations often obscure in a thin and polished section.

Microscopic work, no matter how detailed, is at best qualitative and reliable only to the extent that the small ore sections being studied are representative of the orebody. This, too, is the time to arrive at a mutually agreed definition of terms used by the geologist and metallurgist. For instance, the geologist classifies typical ore by its origin, structural relationships, and mineral composition. Typical ore to the metallurgist, on the other hand, is ore from which the mineral values can be effectively recovered by a predetermined method at a fixed treatment rate and within a narrow cost range.

The Exploration Stage: During exploration, cooperation between the geologist and metallurgist should take a concrete form and the foundation should then be laid for the ensuing even closer coordination of mining and metallurgical planning. As the objective of exploration is to evaluate the deposit for development and exploitation, it is not enough to determine that the deposit is large enough, of sufficient grade, and mineable. Before exploration is completed, a definite conclusion must also be reached as to whether the ore can be effectively concentrated, by what means, and at what approximate cost. The metallurgist must have help of the geologist in obtaining samples of representative ore for testing and comparative evaluation.

Flotation research in progress during exploration would be devoted to batch testing of individual and composite samples to delineate the influence of va-

riations in ore grade, nature, and complexity on recovery and grade of the copper concentrate and by-product minerals such as molybdenite and rutile. It is essential that the general outline of the metallurgical flowsheet be decided on at this stage so that approximate milling costs can be estimated. It is a truism that the more complex and variable the deposit, the greater the importance of early and thorough metallurgical testing, even if fully representative samples are not available. Furthermore, it is the conviction of most metallurgists that facts concerning metallurgical response of the ore to treatment are more important than the mineralogical nature of the ore as a factor in guiding exploration. Moreover, if the metallurgical facts are not properly considered, exploration may be either inadequate or misdirected for obtaining data on recovery, grade of concentrates, milling costs, and smelter returns which are essential for deciding on the feasibility of developing and exploiting the deposit.

In making an evaluation of the ore during exploration, the metallurgist will be primarily concerned with testing enough different samples to insure exploring the effect of ore variables. He must be content with samples available, as it is seldom practical to carry the exploration work beyond determining that the deposit is now worth exploiting. The chief questions to be answered by the geologist, therefore, deal with availability of samples and their representativeness with respect to different types and grades of ore likely to be encountered in the deposit. The type of sample and its history are of paramount importance to the metallurgist. Test pit and core samples are ideal, but churn drill sludges, flocculated and dried for assaying, are seldom worth testing. If churn drilling is used, the complete sludge should be filtered and tested immediately to minimize adverse effects of oxidation and alteration of the sulfides. A cross comparison between core and sludge samples generally gives an adequate correction factor that can be applied to samples from different parts of the orebody to obtain relative information for metallurgical planning.

In addition to information on the extent and nature of the deposit, the geologist should have information on the availability and location of water resources for the operation.

The Mine Development Stage

Once the decision has been made to develop the mine, metallurgical research must be redirected in purpose and scope. Emphasis is shifted from consideration of the broad aspects of winning the valuable metals from the hypothetical ore to a detailed investigation of the unit operations and processes that constitute an operating mill treating a specific ore. At this stage the basic flowsheet usually will have been decided upon and the design started. What remains is an accelerated search for refinements and modifications to improve recovery and grade of product, to resolve anticipated and unanticipated processing problems, and most important to lower overall processing costs. Small-scale, continuous pilot plant tests on one or more blends of ore from development of the mine are required to formalize the flowsheet, test alternative equipment, and obtain operating data for mill design. Attention also will be given to recovery of byproduct minerals not possible in preliminary bench scale tests.

At this stage of the project, all technical problems should be mutual. None of the three groups should be required to work in even a partial information vacuum. Since the mining method chosen will affect the character and grade of ore to be processed, the metallurgist wants to know not only what is being planned and done in the mine, but also why. Similarly, the mining engineer should concern himself with understanding the metallurgical problems and how these are magnified or minimized by fluctuations in the ore supply from the mine. At this time, too, a mutual study by geologists, mining engineers, and metallurgists must be made of the future reserves that are to be processed when the immediately available and most easily treated ore has been exhausted.

Information needed by the metallurgist

1) What are the possible alternative methods and costs of mining and their influence on dilution and grade of ore with respect to the principal mineral, byproduct and accessory minerals, and gangue minerals? Excessive dilution in mining is unwanted, as it actually costs more to process country rock than ore.

2) What effect will the method of mining have on the physical nature of the ore with respect to the proportion of fines, oversize, and moisture content?

3) Will the method of mining add to the complexity of ore treatment and thereby increase milling costs?

4) The mining program with respect to scale of initial and subsequent operation and length of time the ore remains in stopes, is in transit, or is exposed to weathering, and alteration of reactive minerals will have a profound effect on the size and metallurgy of the mill.

5) Information also will be needed on the range of hourly, daily, weekly, and seasonal fluctuations of the ore that must be milled with respect to tonnage and grade, types and complexity of the ores, and variations in physical characteristics.

6) Intelligent planning demands an answer to what provisions the metallurgist should make for handling lower grade, more complex, or different types of ore within 5, 10, or 20 years to take advantage of larger scale operations and technologic advances. It may be impossible to predict what type and grade of ore will be encountered as mine development progresses, but an intelligent guess is better than no information at all.

The mining engineer's answers to these questions will cover the essential data needed by the metallurgist to plan and evaluate the metallurgical research.

Summary

It is the metallurgist's responsibility to point out where the pitfalls lie in milling and smelting so that the geology and mining departments can be on the watch for ore changes that are bound to occur as mine development progresses. No matter how detailed the plans or how sincere the spirit of cooperation among the three technologic groups, if a poor sample is used for the final process studies, the result probably will be disastrous. The metallurgist will be no less the culprit than the geologist and mining engineer if he chooses to disregard their advice and counsel.

●●● *symposium*

Cooperation for Metallurgical planning

Geologist-Metallurgist Cooperation in Porphyry Copper Exploration

- Drill Sampling
- Preservation of Samples
- Laboratory Tests
- Pilot plant operation
- Microscopic Study
- Geological Record-Keeping

by Kenyon Richard

IT is apparent that metallurgists could have designed and operated concentrators more successfully if they had been supplied with better production samples and descriptive data on the character of mill feed. Geologists and mining engineers can provide better data than in many past instances. To facilitate this exchange the metallurgists should define the relative importance, as well as the nature, of this desired information. Some of this information can be supplied only at high cost. This discussion will consider those areas in which there can be improvement in the mutual appreciation of technological limitations within each field.

Once a deposit has been found, exploration becomes specific. Usually this is in the form of a preliminary drilling program with a limited budget intended to evaluate the possibility that an orebody occurs within the deposit. If this preliminary work gives encouraging results, the exploration program is expanded. It is continued as long as results are favorable, the end point being thorough definition of shape and tenor of the orebody.

Beginning with the first drillhole there is a critical period during which the program is subject to momentary abandonment. This situation exists because of geological and environment conditions peculiar to each deposit. It essentially is a problem for the geologists because eventually their interpretations should resolve the paramount uncertainties. During this early critical period the metallurgists seldom have occasion to perform test work or to consult with the geologists except on a casual basis.

KENYON RICHARD is Chief Geologist for American Smelting and Refining Co., Tucson, Arizona.

Beginning with the first drillhole there is a critical period during which the program is subject to abandonment.

However, they should be kept informed on the general character of mineralization when an unusually difficult metallurgical condition becomes apparent. There is a special case in which the metallurgists probably would have to participate actively almost at the beginning of specific exploration of a porphyry copper deposit. If a high proportion of the copper is present in oxidized form, the metallurgy might be complicated and possibly too costly. This special problem must be solved early, as it could have a material effect on continuing the exploration program, and actual metallurgical testing should be undertaken as soon as it is possible to obtain samples.

In other words, during this critical period money should be spent *only* to gain information that will contribute directly to the ever present problem of whether the exploration program should be abandoned or continued.

During the critical period it will not be definitely known that metallurgical testing will ever be needed. Nevertheless, in carrying out the exploration work, it should be assumed from the beginning that metallurgical test work eventually will be done. This involves two important procedures: 1) preservation of samples and 2) geological record-keeping.

Preservation of Samples. Although early exploration samples are taken primarily for assay, any material that might eventually contribute to solution of metallurgical problems should be saved. This includes material that could be of use in microscopical examinations for metallurgical purposes, as well as ore that could be used directly in metallurgical testing. For example, throughout one of the recent drilling programs a split of the diamond drill core was crushed and a portion pulverized for assay. The crushed rejects were saved and used for laboratory metallurgical testing, Also, a sulfide concentrate was produced by panning a portion of the pulverized material from each core interval. These concentrates were composited, and bakelite briquettes were prepared representing 20-meter intervals in practically all drillholes. The polished briquettes were studied under a reflecting microscope and textures and pyrite-chalcocite-chalcopyrite ratios noted. This was done primarily for geological purposes. However, these data were helpful to the metallurgists during pilot testing. More important, the briquettes are permanently available in case they are needed to solve metallurgical problems arising during full-scale milling.

Geological Record-Keeping: The amount and kind of detail recorded for drillhole samples vary according to the geological character of the deposit. This study is made primarily for current guidance of the exploration program. However, these detailed geological logs will serve to answer many of the metallurgists' questions regarding mineralogical character of the ore, and the logs, of course, always should be available to them.

In the general case the need for metallurgical testing arises at the end of the critical period, that is, when it has become evident that the deposit has a definite commercial aspect. When this stage has been reached the problem will arise as to which ore types should be assembled into samples for metallurgical experimentation. At this point the metallurgists and geologists should understand each other in regard to the following three basic concepts: 1) All orebodies are nonuniform in terms of geology and of metallurgy. Development of over-simplified ideas about the geology of porphyry copper orebodies is a common and serious error made by geologists and mining engineers as well as metallurgists. 2) *Average ore* is a meaningless term unless it is defined, first, according to a specific production interval such as a day, a month, or a year, and second, according to mineral composition as affecting metallurgical character. 3) Important geological subdivisions of an orebody may have no metallurgical significance; conversely, mineralogical details that may be of practically no interest from the geological standpoint can be highly critical metallurgically.

Assuming that the foregoing conditions are mutually understood, the metallurgists should review the detailed geologic logs and all geological reports containing mineralogical descriptions. They should then set up rough specifications as to the probable metallurgical response of the various types of ore, noting particularly the more refractory materials. Using these specifications the geologists and mining engineers can determine the relative significance of these various types in terms of tonnage and grade, and it then can be decided which ore types should be tested. Obviously, this all requires very close cooperation among the metallurgists, geologists, and mining engineers.

Presumably the first metallurgical work should include separate laboratory tests on those types of ore which are more important tonnage-wise. If there has been occasion to penetrate the orebody with mine openings during the early period of exploration, the test samples should, of course, be taken from these workings. However, this situation is rare. Usually the only source of preliminary test material is core or churn drill sampling. To provide data on different types of ore, core samples should be composited in mineable units matching the metallurgists' specifications. These samples will be relatively uncontaminated, but they will comprise a small volume of material. Churn drill sludge samples always are contaminated with grease. Notwithstanding, they sometimes are used for preliminary laboratory tests if no other material is available, the results being indicative of variations among ore types.

When the orebody has been delimited by drillholes, or nearly so, the problem of obtaining bulk samples for pilot mill testing will arise. Laboratory testing will have indicated metallurgical variations among the ore types, and the geologists and mining engineers can provide preliminary estimates of tonnage and grade of those types which the full-scale mill will handle during successive mining periods. By putting all these data together it can be determined where shafts should be sunk or adits driven in order to obtain the desired bulk materials. Depending on topography and depth and shape of the orebody, this work may be expensive. It probably will be impractical to obtain representative bulk samples of all types of ore, particularly those found only in the deeper parts of the orebody. The extra expense of obtaining samples of deep ore will have to be balanced against estimates of the effect this material could have on basic mill design and against the allowable error in operational metallurgy. In this situation management cannot expect the metallurgists to be exact, and the amount of money spent on acquiring test samples should be commensurate with the metallurgists' ability to make estimates within acceptable limits of accuracy. As an example, pilot mill testing of one porphyry orebody has involved only chalcocite ores, whereas large tonnages of deep-lying chalcopyrite ores have been outlined by

drillholes. Laboratory testing of core samples of these deep ore types indicated that the metallurgical uncertainties were relatively minor, and the cost of obtaining bulk samples would have been too high to justify the work.

During pilot mill operation the geologists should provide the metallurgists with mineralogical descriptions of the mill samples, particularly as to how representative they are in terms of full-scale production time intervals. The mining engineers should acquaint the metallurgists with details of mining and handling the test samples, so that such features as contamination, dehydration, and oxidation can be taken into account. Indeed, the metallurgists should be in position to insist that certain changes or precautions in sample handling be instituted if desirable from their standpoint.

Throughout many porphyry copper deposits the boundaries between ore and waste and between good ore and lean ore are gradational. However, in some of these orebodies changes in grade and metallurgical character are sharp and close-spaced. For orebodies of this kind, the geologists' ore reserve estimates and the mining engineers' operating layouts both should be made in a manner that will provide general information on the extreme changes in ore types in relation to production time intervals. It is important to understand, however, that through large portions of most orebodies drillhole sample information is too widely spaced to permit accurate predictions of these extremes during small production time intervals. As an example, the existence of post-ore dikes with possible refractory metallurgical characteristics is known in one undeveloped orebody. These dikes probably are too narrow and too wide-spaced to be mined selectively by power shovel and their distribution is imperfectly defined by drillhole intercepts. As a result they have not been taken into account in ore reserve calculations or metallurgical planning, although on rare occasions this dike material may cause trouble in the mill.

To extend this line of reasoning further, certain important metallurgical characteristics may not be evident to the geologists. This may be due 1) to the geologists' lack of knowledge of the metallurgical significance of certain minerals or obscure textural relationships, 2) to the difficulty of describing the physical character of a rock in terms the metallurgists can translate into the rock's probable action in the mill circuit, or 3) to the fact that the drillhole samples the geologists classify are not representative of the ore.

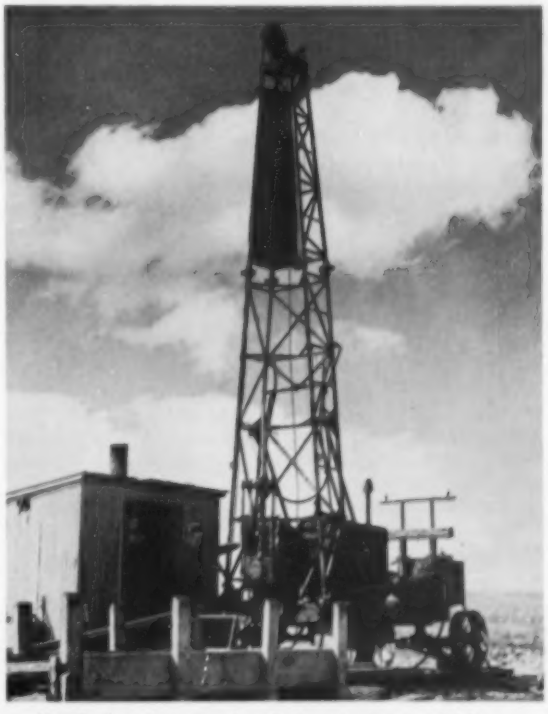
Case 1 above is not very important and can usually be avoided if the geologists have sufficient experience. Regardless of the quality of their education, young geologists must have on-the-job guidance, by metallurgists as well as senior geologists, in order to appreciate those geological features that may be of concern to the metallurgists.

Another aspect of case 1 is the matter of microscopic study. For two principal reasons geologists usually resort to thin-section and polished-surface work during exploration: first, to differentiate rock and mineral types as a direct aid in the guidance of the exploration program, and second, to detect any special mineralogical characteristics that could result in systematic errors in drillhole sampling.

Case 2 above refers particularly to the action a rock will have during crushing, grinding, and settling. It is entirely impractical to drive enough underground openings to be certain that every important physical characteristic is known for each ore type. From study of drillhole samples the geologists can estimate and describe these physical characteristics only in generalities. It is impossible, therefore, to foresee many of those common milling problems related to rock hardness, abrasiveness, and other characteristics. Management and metallurgists must expect that operating difficulties will arise from incomplete knowledge of physical characteristics of some ore types.

Case 3 is an extension of case 2, but it refers more particularly to churn drill samples. The churn drill is popular in exploration of porphyry copper deposits because it provides a relatively large and accurate sample for assay. However, the churn drill sludge sample gives incomplete data on geology. Usually the small fragments and grains are examined megascopically and microscopically after the slimes are washed from each sludge sample. All recognizable mineralogical features are noted, or should be, but much evidence on mineral textures, structures, and the overall physical nature of the rock is lost. Therefore, in the case of orebodies largely explored by churn drilling, the geologists are unable to obtain sufficient information to answer many of the metallurgists' questions about the physical characteristics of the ore.

The questions that the metallurgists would like to have answered by the geologists fall into two broad categories. The first includes those that can be answered if there is technological cooperation. The second category consists of questions that cannot be answered by the geologists unless exceptionally high-cost samples are obtained. This involves a balance between necessity and cost. Although the geologists and mining engineers can assist in this matter, it is chiefly the metallurgists' responsibility to arrive at this balance.



Usually the only source of preliminary testing material is core or churn drill sampling.

Metallurgical Planning in Connection with Lead-Zinc or Lead-Zinc-Copper Mines

by M. B. Kildale

INITIAL concepts of a new mining enterprise may be due to a geological idea, development of a new and more economical mining method, or improved metallurgical treatment. On whatever basis exploration and development work is undertaken, it is now generally recognized that all three branches of the mining business represented on this panel play vital roles in bringing any new mining project to fulfillment, and each branch can contribute much to the other two branches.

Speaking as a geologist, the author would like to emphasize that cooperation works three ways—and that in most cases data that can be supplied only by the metallurgist and mining engineer are of great importance to the geologist even during the exploration and development stages of a projected mining operation. For example, it is usually the function of the geologist to calculate the ore reserves that form the basis for all operational planning. To calculate reserves the geologist must know what constitutes ore or what is likely to be ore at that particular deposit, and the determination of this fact is dependent both on extraction costs and on the value of the recoverable metal as contrasted with gross metal content. Any decrease in estimated mining costs or any improvement in metal recoveries as metallurgical testing progresses may lead to an increase in ore reserves and thus affect the projected scale of the whole operation. Close cooperation from the start of development work may pay big dividends.

In actual practice, although the general principles of cooperation are rarely questioned, problems often arise as to just how the highest degree of cooperation can be obtained in one particular operation. With reference to lead-zinc or lead-zinc-copper operations, F. W. McQuiston has mentioned some of the technical problems confronting the lead-zinc-metallurgist and has listed ways in which the geologist and the mining engineer can be of help in metallurgical planning. Following up his summary, the following discussion outlines those areas of cooperation lying within the province of the geologist.

As mentioned above, it is the job of the geologist to supply the metallurgist with figures on the ore reserves of the deposit being developed. These figures of course are subject to frequent changes as development work progresses, and through continuous communication the metallurgist should be advised of all such progressive changes. Furthermore, the geologist must advise the other departments not only as to the amount of proved or blocked out ore but should also give his estimates of probable and possible ore—and should make sure that the metallurgist understands just what is meant by probable or possible or *indicated* ore. Estimates of possible ore are often a matter of geological judgment but are also vital in operational planning. For example, in the case of lead-zinc replacement deposits in limestone, it is usually impossible to block out the total mineable tonnage before the mine is put into operation. There are many instances here in the West of lead-zinc replacement mines that have had lives of 15 to 50 years, where the blocked out reserves have never been equal to more than one to two years of production. In such cases the planning of a mill cannot be based solely on calculated reserves of proved ore, and the geological inferences as to ultimate life of the deposit must be made clear to those doing the metallurgical planning.

Replacement deposits of lead-zinc or lead-zinc copper ore are characterized by great variation in types of ore occurring within one deposit or one ore zone. They characteristically show intermixing of oxide and sulfide types and are susceptible to zoning variations, vertically or laterally. In cases showing such variations the ore reserves of the various types or classes should be calculated by the geologist for the benefit of the metallurgist and mining engineer, and in his subdivision of reserves the geologist must be governed largely by the metallurgical characteristics of the ores. In some cases division merely into oxide and sulfide ore may be sufficient, but if enough variations occur within these two general types, further splits must be made. For example, at the Darwin mine in California a large percentage of plumbojarosite occurs in one portion of the oxidized zone. This was noted and recorded by the geologist, but since the ore was of average grade, no significance was attached to this observation. However, when this ore was later mined and delivery to the mill began, a drop in percentage recovery of lead was noted and the metallurgist began to make inquiries. It was then determined that the plumbojarosite was a low recovery mineral "requiring a greater amount of sulfidizing reagent and causing slimes which did not settle readily." Although the tonnage of this type of ore was not large, this trouble could have been avoided if the metallurgist had been informed of this type of ore and if the ore had been tested in advance of mining. Determination of component minerals in oxidized zinc ores, often requiring microscope studies, is also often important in working out milling methods and determining zinc recoveries.

Furthermore, many lead-zinc or lead-zinc-copper deposits are first developed entirely in the oxide

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zone and initial production is all from oxidized or partially oxidized ores. However, it can often be predicted that these will grade into sulfide ores at depth, and such geological inferences should be considered in planning future methods of concentration.

F. W. McQuiston points out that obtaining samples for metallurgical test work is an important duty of the geologist or the geologist-metallurgist team during the development stage of an operation. And here the two departments should definitely work together as a field team, taking trips through the mine. The geologist may be the first to recognize the various types of ore in place—or it may be that variations in mineral composition not readily recognized by eye may be detected during test work in the laboratory. In either case the metallurgist should tell the geologist what samples are needed, with information as to type, size of sample, and purpose for which the sample is intended. Proper sampling is the responsibility of the geologist.

During early stages of exploration sampling is usually done solely to determine gross metal content but the geologist should also keep in mind the needs of preliminary metallurgical work by saving the rejects from assay samples for metallurgical purposes. Samples showing mineralogical variations or various degrees of alteration can be collected to show to the metallurgist.

A geologist who is aware of metallurgical problems will realize that the physical characteristics of lead-zinc or lead-zinc-copper ores are also important from the metallurgical viewpoint, and at the request of the mill superintendent, he may have to classify ore blocks on that basis. For example, ore shoots consisting of scattered sulfides in hard garnetized limestone will require a very different crushing setup from that used for a 50 pct replacement in soft shale. And a coarsely crystalline mixture of galena and sphalerite will be given a different grinding treatment than a very fine grained, intimate mixture of lead, zinc, and copper sulfides. If such variations occur within one deposit the metallurgist has problems, and in solving them he can be greatly aided by information supplied by the geologist and by representative samples obtained under joint supervision.

To summarize, the three *major* contributions to metallurgical planning which lie within the province of the geologist are: 1) information on blocked out and indicated future tonnage, within the deposit to be exploited; 2) information on ore types within the deposit, including mineral composition, physical characteristics, and distribution, to facilitate calculating of ore reserves for each type as differentiated on the basis of metallurgical properties; 3) supply of representative samples, in conjunction with the metallurgist, for test work in the metallurgical laboratory.

There are other fields in which the geologist may aid in metallurgical work. He can predict probable changes in the ores on deeper level or in other undeveloped portions of the ore deposit. He can report the occurrence of gangue minerals, such as barite, in the ore. He can also take note of minerals containing desirable accessory metals or undesirable impurities, such as bismuth, arsenic, or antimony, and can supply data on the association of silver with certain minerals as galena, or association of gold with pyrite, arsenopyrite, or sphalerite (as indicated by assays).

Mining Engineer's View of Cooperation

by A. G. Kirkland

ALTHOUGH the geologist and mining engineer customarily work together in dealing with common problems, there may be room for greater cooperation between the two technical departments responsible for finding and mining the ore and the metallurgist whose responsibility it is to obtain maximum value from the ore.

In connection with the processing of a base metal ore from an underground operation, the metallurgist must have from the mining engineer and geologist the answers to many questions before he can intelligently and effectively design the metallurgical plant and process. In some cases, by the time the mining engineer enters the picture, the geologist may have concluded the examination stage and may be well into the exploration stage. The metallurgist, through cooperation with the geologist, may have received his first samples of the ore from drill core, channel, or bulk samples and may already have conducted his preliminary research as to how to treat the ore. He may even have worked out a tentative flowsheet.

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The geologist may have discovered, or had brought to his attention, what at first glance would appear to have encouraging economic possibilities. In fact, it may look so good that he launches into an examination and preliminary exploration program. Stimulated by the results of his preliminary efforts, he brings it to the attention of the mining engineer. Only after the following items can be satisfied in favor of a capital venture can the mining engineer consider profit possibilities:

Is the market favorable to the product?
Is transportation available, and at what cost?
Is an adequate water supply available?
Is power available, and at what cost?
Is labor force available?
Is the climate suited to reasonably efficient operation?

Some of these matters to be considered do not present the same problems they did a few years ago. However, each of them is of sufficient importance to warrant attention from the mining engineer before the geologist becomes over-optimistic about the potential of his discovery.

Cooperation between geological, mining and metallurgical departments in any company is of course not something that is exercised at the discretion of the personnel in any one of the departments. It is up to top management to encourage and maintain this necessary cooperation.

The first question the metallurgist will ask the mining engineer will concern initial rate of production and how soon the mine will be in a position to produce such tonnage. In any type of orebody, the stockholders and management cannot and should not wait, nor is it practicable to wait until the outline of the entire orebody, or bodies, has been fully disclosed before designing the processing plant. When enough ore has been indicated to justify a mining and milling plant, the push for production is on.

Metallurgical research work should begin when the first samples have been taken and the program stepped up as exploration reveals increasing tonnage and information. In most cases the treatment plant is not erected until enough tonnage is indicated to justify its cost—the cost of a mining plant plus a profit. There have been cases, however, where a mill was put up with scarcely any knowledge of tonnage or grade.

In some cases, planning for initial production rate depends on economic urgencies and on the amount of information as to tonnage and grade the geologist has been able to gather. Some orebodies are so massive that a huge tonnage is revealed by each drillhole, whereas in some vein-type mines with complicated geology a much longer and costlier program of exploration is necessary. Ultimate rate of production may be visible on the horizon even before final plans have been laid by the metallurgist. In most cases, however, production increase is possible only after a long period of exploration and development in which ore is found at a much greater rate than it is being mined.

Since the metallurgical research work, the design of plant and process, and the mill or smelter construction are usually done under pressure of time, early cooperation between the geologist and the mining engineer on one hand and the metallurgist on the other is extremely important. For each mine there is probably an economic optimum rate of output, usually based on size and nature of the orebody, method of extraction, and facilities for handling ore. The metallurgist knows that economies will result if provisions for later expansion can be designed into the first stage of plant and process. Undersized mills, constructed with no provision for future expansion, have soon been overcrowded with tonnage to a point where maximum recovery, cost, and profits have been sacrificed. Where no provision for convenient expansion and uninterrupted production has been made in the research and early planning, a plant shutdown and added cost is inevitable. A good part of this unnecessary additional cost may be avoided by greater cooperation between the geologist, the mining engineer, and the metallurgist.

Before designing the flowsheet the metallurgist should know something of what can be expected from the mine regarding consistency of quantity and quality of mill feed and its degree of hardness and fragmentation. The mining method and its flexibility and the capacity for storing ore in stopes or pockets will have a bearing upon uniformity of daily output. If the daily mine output is unavoidably erratic, provisions must be made in the mill or crushing plant for greater storage capacity to handle surges. Leveling off of production surges is possible by providing larger pocket capacity underground. The metallurgist should be consulted regarding the feasibility of underground crushing.

If the grade of the orebody is uniform, the metallurgist can plan for maximum recovery. Geologist and mining engineer should discuss with the metallurgist the possibility of unavoidable variation in the grade of mill feed. The mining engineer or operator is far from unhappy when the grade of production suddenly shoots upward and naturally feels that all those in the management level of the operation feel the same way. All concerned will be happy except the metallurgist who has designed the flotation section of the mill for uniform grade of ore. Capacity and recovery should be balanced to a maximum point and the balance maintained. Variation in one sometimes affects the other. If the metallurgist has been informed by the mining engineer that he can expect an increase in grade he can provide for flexibility or low cost expansion. The geologist and the mining engineer may know well in advance that as the mine workings extend laterally or in depth, the predominance of one base metal in the ore may give way to another.

When the geologist has submitted his estimate of tonnage, grade, outline, and character of sufficient ore to justify a mining and treatment plant, the mining engineer is primarily concerned with the mining method to meet the scheduled production rate at the lowest possible cost per ton delivered to the mill bin. After a thorough study of all the information available, the mining engineer may conclude that a low cost method of stoping such as shrinkage or caving is applicable. By this method a considerable tonnage of broken reserve is built up in stopes. If the interval between breaking the ore and delivering it to the mill is a matter of months, under certain conditions the exposed surfaces of broken ore oxidize rapidly, resulting in poor recoveries if preparations have not been made by the metallurgist to treat such ore. The mining engineer, or operator, may make a fine showing on cost per ton delivered to the mill, but unless he has cooperated with the metallurgists this showing may be at the expense of efficiency in the mill or smelter. The metallurgist may conclude that the ore is amenable to heavy media separation. If so, he should immediately advise the mining engineer, who is then permitted greater latitude in a low cost, less selective method of mining.

The mining engineer, of course, does not lay his plans for mining solely with the metallurgist in mind. His responsibility is to deliver the ore to surface at the lowest possible cost per pound of contained metal. It is up to the metallurgist to conduct his research, design the process, and operate the plant at the least possible cost per pound of contained metal and to achieve maximum recoveries.

The geologist and the mining engineer have the advantage over the metallurgist in that they usually know in advance occurrences underground that could have a bearing on results in the mill or

smelter. They will know, or will have estimated, the type, grade, and amount of dilution to be expected, the degree of ore fragmentation, and the circulation of air through broken reserve that may cause oxidation or when changes in the type and grade of mineralization occur as mine workings increase in depth or extend laterally. The metallurgist must know all this in advance from the geologist or mining engineer if he is to plan and operate to win maximum returns from the ore delivered to him.

Cooperation between mining engineer and metallurgist should not end when the metallurgical plant and process have been designed but should continue during the productive life of the operation. An understanding of each other's problems will make way for easy cooperation. An occasional visit underground by the metallurgist and trips through the mill or smelter by the mining engineer, or operator, will not only be educational but will foster good relationship between the two departments.

Reno Sales:

"Full Cooperation Between Departments Is a Must Policy for Mining Enterprises"

THESE authors who have covered the subject so thoroughly have abundantly shown that from the beginning of a prospecting venture and up to and through the development and final operating stages, full cooperation between the geological, mining, and metallurgical departments, is truly a must policy for mining enterprises.

The papers deal with so-called porphyry coppers and lead-zinc ores, but cooperation is just as important in any mine producing valuable metal. Make no mistake about that. Saving more copper, zinc, or lead and at a lower cost is a highly desired objective. It is true that in the case of copper the chief source of that metal in the western hemisphere has been the *porphyries*. As we look to the future, we are facing the problem of better metal recoveries from our known ore reserves, and also that of beneficiating even lower grade ores. Closest possible cooperation between technical departments so strongly urged here today will go far toward the solution of these important problems.

The authors have urged, and rightly, that to increase metal recoveries from ores the metallurgist must go far beyond the use of miniature laboratory test plants. We may well draw the inference here that in the past some mills and concentrators constructed on the basis of knowledge gained only through test plant operations were built with insufficient detailed knowledge of the physical and mineralogical composition and variations in the ores being tested. And I think we may say in truth that we can put the blame for this upon the lack of understanding and cooperation between the departments of geology, mining, and metallurgy. The fact is, the geologist was a pretty rare bird in most mining companies until the more recent years, and he wasn't supposed to step inside the concentrator, or ask questions of, or offer advice to, the metallurgist, or even to the miner.

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We can all understand that it has been impracticable for those presenting these interesting and valuable papers to follow through on the question as to what kind of help the miner or geologist can give to the metallurgist. Most of the authors refer frequently to the importance of help in prospecting and exploration stages to enable the metallurgist to do preliminary testing in the laboratory, or up to the pilot plant stage, in order that the plant finally built will have not only the best possible flow sheet but also the proper daily capacity. But with the plant in operation it rarely happens, if ever, during the life of the orebody that the metallurgist is not seeking out new processes, or improvements on the one already installed, in order to increase metal recoveries and lower costs, and it is here also that a study of the mineralogical and physical make-up should play an important role.

As has been said, one important job is to make the maximum recovery of metal, that is, from the viewpoint of profit. There was little mention of the loss of metals. After all, mining companies, especially those in the fabricating business, are interested also in the continuity of flow of metals to their plants over the longest period of time possible. So is the public and the nation interested that there be no loss of valuable metals.

Speaking of metal recoveries, I am reminded of the occasion where a geologist was making intensive studies of the mineral compositions and complexities of a certain ore, in close cooperation with the metallurgist whose final plant was not making as good recovery of metal as had been obtained in the smaller pilot plant. At a later date, to the surprise of the geologist, the metallurgist advised that no more help was needed because the new finished plant was making a slightly higher recovery even than had been made in the pilot plant. Taken by surprise, the geologist asked, "Are you going to stop at the percent recovery you are now getting? Why not try for a 100 pct recovery?" To me that makes sense. Any improvement in metal recovery

FROM PROSPECT TO PRODUCTION is a long path made smooth only by cooperation in exchange of ideas and information between three specialties: Geology—Mining Engineering—Metallurgy. The end result of this cooperation: today's large scale integrated mining operations.

might be well worth going for. Well, let us see if it is, from the dollars and cents viewpoint.

It doesn't take many of our present day copper mines to add up to production of 200,000 tpd, or say, 70 million tons per year. Assume that by full cooperation of all departments, and by research, we could recover 1 lb more of copper per ton of ore without substantial capital expense. It would mean, at a 40¢ market price, or let us say a net gain or profit of 25¢ per lb, $17.5 million per year. This would be realized plus a saving of 70 million lb of copper. That surely is worth a lot of cooperating in anybody's language, and this increased profit would take care of a lot of study and research programs.

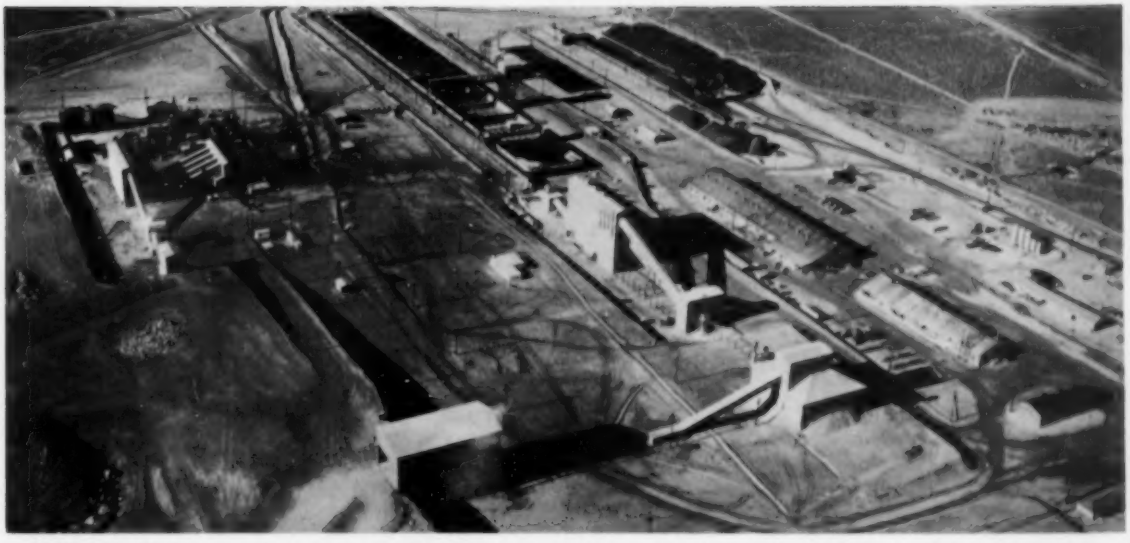
Most of the authors of the papers stress cooperation between geologist and metallurgist in the prospecting and exploration stages of a property. The mine operator was brought in as soon as it became evident that a metallurgical plant was justified. Because of the fact that orebodies of the "porphyry" copper type are being stressed in these papers, it might be well to note that the metallurgical treatment of the chalcocite blanket type of porphyry has become more or less standardized. But we have to deal with wide variations from the chalcocite type of ore, where calcopyrite and bornite are the chief copper minerals. And there are the mixed ores, the headaches of the metallurgist, and most low grade copper ore contains a little unrecoverable copper in oxide form, or a small proportion of water soluble copper. Oxidized copper ores requiring some kind of leaching process are not uncommon.

Among these wide variations in mineralogical make-up of copper ores, the problems of the metallurgist are likewise variable from relatively simple to extremes of complexity. With the more simple ones requiring more or less standardized flowsheets, the question arises, of course, as to whether, with the aid of research laboratory, any further worthwhile recovery is possible. The copper lost in the tailings may be too locked up mineralogically to be recoverable at a profit. But the attitude of the management should be never to admit defeat.

In the case of the more complex sulfide combinations of copper, zinc, silver, gold, and molybdenite, *geological laboratory* study should be very helpful from the first prospecting work through to the completed treatment plant, and for as long as the orebody lasts. And there is the problem of winning from the ore every metal of commercial value. Some of them may be in the so-called rare metal group, occurring in the ore in minute quantities but of high market value. Only the well equipped research laboratory can cope with the problem of their identification and recovery. The motto of the metallurgist should parallel that of the butcher of the pig, which is, "We save all of the pig but the squeal."

So, I was quite surprised to note that not one of the papers urged, or made more than passing comment on, the pressing need of research laboratories, either geological or metallurgical, for without these facilities a severe limitation is placed upon those endeavoring to get the best possible results from their beneficiating plants. The statement has been repeated in nearly every paper presented that cooperation, particularly between the geologist and metallurgist, is absolutely necessary for the proper design of a plant, or for working out a concentrator flowsheet, and so on. But, how far can the geologist or metallurgist go toward a detailed and correct understanding of the complexity of mineral aggregates, as to their good or bad effects on processes, or on the problem of determining the why of metal losses, without properly equipped research laboratories? No further, I suspect, than has been the practice over past years. Here cooperation between the geologist, miner, and metallurgist cannot do the job that should be done, unless the tools needed for improving results are provided. These research studies should follow every step in the operation from the crusher through the mill and smelter, and finally to the slag dump and tailing pond.

The mining industry must awaken to the fact that research may mean its salvation, not only as an offset to increasing costs but as a measure for add-



OPPOSITE PAGE: The open pit at Anaconda's operation at Weed Heights, near Yerington, Nev. ABOVE: General view of Anaconda's entire plant at Weed Heights. The treatment plant (also shown above) can be seen in the distance, upper center of photograph.

ing to reserves of ore. In past years, the loud pedal has been put on research involving the marketability of products, and on the selling end, with far too little attention paid to the mineralogical composition of ores, to mining, and to the recovery of metal from the ores.

Little has been said today of coordination. The question arises, who in the organization is to take the responsibility and see to it that any plans of cooperation between the geologist, miner, and metallurgist are efficiently carried out? We all know that the heads of any of our technical departments are busy individuals with their work cut out for them every day. In many cases the mines and treatment plants are five miles apart. Furthermore, department heads are not always inclined to accept suggestions or advice from any other department head. Coordination, a very vital factor, is the duty of the general manager of the enterprise, who, for best results, should have a good working knowledge of the techniques involved in the various departments. However, the great importance of the kind of cooperation between the manager and technical departments being urged here today must be recognized also by the officers and directors of the enterprise. Otherwise the manager may find himself behind a road block whenever changes requiring capital expenditures are involved.

Therefore, from the top company officers down through the entire organization, the magic words are understanding and cooperation, and may I say, research. I think I can predict with some degree of confidence, however, that unless the manager and the company officers are completely sold on the value of the very excellent ideas of cooperation expressed at this meeting, that is, to the point where they will be put into practice, this symposium will have been for naught.

In summary, we must recognize that these authors are properly advocating a definite and most desirable program for improving the practice of mining and metallurgy. Their efforts should be commended highly by all who are interested. I am sure that their recommendations will receive the most serious consideration. What they propose marks a progressive step in cooperative management of mining enterprises, and a clear advancement over earlier practices. But we must not stop here. I dare say that in the case of any complicated flowsheet involving the course of an ore from the mine to the finished products, no metallurgist, miner, or geologist will admit that there won't always be much room for further refinements, both practical and scientific, even after today's recommendations are put into practice. How then can we attain these additional refinements and improvements marking the next step toward our final goal? The answer is, continued cooperative study of every phase of the operation with the aid of properly designed and equipped research laboratories.

One further comment does not seem out of place. Almost without exception, the authors stress cooperation in that period between the prospect stage and the finished mine and plant as a going enterprise. The inference seems to be that a lot of new porphyry type of orebodies are yet to be found. I am glad we can look forward to that, but two big problems now facing the low grade mines are 1) how to recover more metal from the ores being processed, and 2) how to put into the profit column the commonly associated lower grade rock. In so many cases, the deposit being worked is surrounded by, or associated with, a zone of lower grade rock, both laterally and in depth. Great improvements in mining methods, mining equipment, and metallurgy have made it possible to go to lower and lower grade ores, and advancements along these lines will continue. In this situation, one of the important possibilities offered to extract more profit from the ores being treated, and to bring into production still lower grade ores, is not only the cooperation stressed here today, but also the establishment of research laboratories through which the cooperation and understanding of the problems between the geologist and metallurgist may be made more effective.

Economic Factors In Cold Weather Operations

by E. B. Spice

ALTHOUGH much may be learned about the economics of cold weather operations by studying successful mining ventures in southern and central Canada, it is the purpose of this article to extend the study by considering only permafrost areas. Geographically this means most of the country north of the 60th parallel. In this area, at the present time, only five Canadian companies are carrying on routine and successful mining operations on a year-round basis. However, there have been some score of unsuccessful ventures, and a number of properties in the development stage show promise of being producers. Problems of cold weather are worthy of study.

It may be pointed out that successful mines in this area face problems of isolation and costly transportation[1] as well as cold weather. To reach valid conclusions, consider the development of a mythical mineral deposit located on the 65th parallel in the heart of the barren lands; problems of isolation and transportation will be ignored.

Exploration and Development

Prospecting: The mineral occurrence has been discovered by a small prospecting party whose active season is no more than four months. Snow covers the ground for the rest of the year. The discovery was made on an outcrop and could be traced for only a short distance. Overburden was frozen and could not be removed by normal hand methods. Thus a preliminary assessment of the ground does not indicate strike length or other economic factors. The normal moves at this point would be test pitting, sampling, stripping, and staking, followed by geological mapping, diamond drilling, and possibly geophysical testing.

Efficient staking usually follows an intelligent interpretation of structure; however, in this climate a thorough examination is impossible before the snow flies and transportation stops. It is not uncommon, therefore, to stake large areas around the initial discovery in the hope of covering any conceivable extensions of interest. This done, the prospectors are taken to more southerly climates for the winter. (Some are retained on salary the year round).

All winter, in the comfort of an office, plans are laid for the coming summer season. At this stage costs are very important, since the extent and value of the showing are unknown.

It has been found by experience that stripping of frozen overburden is very nearly impossible or so very costly that it does not usually figure in an assessment. Electrical geophysical methods are of little use because of the effects of ice in the overburden. Diamond drilling is normally chosen as the first means of evaluation. Some preliminary geological mapping is usually done.

As the season is short, the program must either be limited, or larger than normal crews must be

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assembled. Base and mobile camps must be adequate to make living comfortable during the early and late parts of the season. Fuel requirements are large.

Diamond drilling[2] costs are high even in midsummer, since permafrost makes the heating of drilling water essential. Generally drilling is limited to 600 ft in permafrost. Contractors will not usually enter in to a fixed price contract under these circumstances. The usual form of contract calls for a fixed price per foot drilled, with extras charged to the owner as follows: fuel oil for water heating, cost of pipe or hose used to supply warm water to drill site, cost of drill rods abandoned because they are frozen in the hole and cannot be removed on an economic basis.

Some drillers use salt or $CaCl_2$ solutions, fuel oil, or kerosene for drilling fluid and arrange for desilting and recirculation of the fluid.

Housing of drill and drillers is customary in the cold weather. Heating of the structure is common and fires are numerous.

Only fully qualified drillers can operate successfully under these conditions, since operating errors can waste time and result in losing a hole and a complete string of rods. Holes must be logged and surveyed immediately after completion; otherwise, ice will prevent access. Drilling is normally confined to the summer months during the speculative stage but is undertaken all year round when the outlining of a promising showing must be expedited. Costs under these conditions are nearly double.

Assuming that diamond drilling has been advisable and that a substantial orebody has been outlined after a year of drilling, underground development is probably the next move.

Open pit operations have not been attempted this far north. However, a large program of this kind is being undertaken at the Gunnar property in the Beaverlodge Saskatchewan area and should prove interesting as a pioneer move.

Collaring of a shaft is the next step. Site choice is outcrop where possible. Overburden sites are exceedingly troublesome and can be prohibitively expensive due to permafrost and foundation problems.

Usually drilling for blasting must be done dry, since air or rock temperatures are low enough to freeze drilling water unless heating of the water and insulating of pipelines is carried out. Recent development of tungsten carbide drill bits has been a boon to dry drilling of shaft collars in the far north. The newer dust-free dry drills may also be of interest for operations of this kind.

The concrete collar is poured, sometimes with $CaCl_2$ added to the mix, almost always with heated water in the mix and with adequate protection for the newly poured and curing concrete.

Headframe construction is unique in that the finished structure generally includes a shaft house of some extent so that future jobs of cage tending, decking, switching of cars, and often dumping, can be done in comfort in a heated and frequently insulated structure.

Water lines to this shaft house generally originate from a well built and insulated pump house at a nearby lake. Provision must be made at the pump suction to keep the immediate area free of ice. This is usually done by installing the suction lines in a crib into which condensate or hot water is run continuously. Pump discharge lines are almost universally above ground, steam traced and housed in pipe boxes with adequate insulation. Almost all types of insulation have been used (sawdust, shavings, fiber glass, asbestos). The cheapest pipe box is wooden and rectangular, with sawdust surrounding the pipes. The top is roofed with tar paper or roofing paper to prevent moisture entering and ruining the insulation. This box is not fireproof and will not stand up if used as a sidewalk. However, such boxes are not unusual.

This article surveys the many ways in which cold weather can affect economics of mine development and operation. All phases of the industry are considered—exploration and development, engineering design, construction, mine and mill operation, transportation, and other services. It is possible to draw the following conclusions: Cold weather always affects adversely the economics of mine operations but, more important, it imposes greater responsibilities on design and operating engineers. Careful planning in the development stage, attention to detail in the design and construction stage, and wise operating decisions can often reduce operating costs to a satisfactory level. Naturally, capital and other preproduction expenses are increased.

The usual prospect shaft will be surrounded by the following structures: hoist house, change house, kitchen, bunk house, boiler house, compressor house, oil tanks, powder and cap magazines (separate), assay office, and powder thawing facilities. These units, if of a temporary nature, are not necessarily insulated, but the cost of heating will be correspondingly high. Some engineers design so as to have most of these facilities under one roof to get more efficient use of heat. However, insurance underwriters look with disfavor on this practice.

Shaft sinking and development proceed in a normal fashion except that permafrost may extend many hundreds of feet below surface, making it necessary at times to heat ventilation air and water.

Open shafts and raises, which are downcast in the winter, must be sealed off completely. For this reason, natural ventilation cannot often be used. Surface dumping of waste and ore from development is difficult. Wet muck freezes to car bottoms almost instantaneously. Bearings on mine cars become stiff and require special lubricants. Shaft sites are usually chosen so that tramming distances will be a minimum and head room for ore and waste dumps a maximum to avoid cold weather difficulties. Waste disposal from bins included in the head frame is practiced but the bins must be heated, which is a difficult task.

Despite these difficulties, it will be assumed that the underground development and diamond drilling have been successfully carried out and that grade and size of the orebody justify construction of permanent mining and milling facilities.

Here the problems are roughly the same as those outlined above except on a much larger scale. At this stage large capital expenditures are justified by the permanence of the installation and the expected savings in operation.

Design and Construction

It will be assumed that the entire program is to be contracted. Choice is limited, since very few consulting engineers and construction firms have far north experience. Competitive bids are nonexistent and fees will be high. In fact, no fixed price work has as yet been done in the far north. Contracts are usually cost plus a fixed fee and the contractor will always protect himself adequately for the contingency of cold weather construction.

Because it is practically impossible to build on or excavate frozen overburden, shaft site and mill site are invariably outcrop. This means that otherwise logical sites must often be rejected.

To conserve heat and to minimize outdoor movement of personnel, head frame, shaft house, engineering and geology offices, administrative offices, assay offices, change houses, ore and waste bins, research facilities, diamond drill core logging facilities, and a host of other services are located in one huge building. The mill building is generally remote and ore delivery is by underground conveyor from an ore pass adjacent to the head frame. Steel, concrete, or concrete block are the usual construction materials. These are not necessarily ideal cold weather materials, but fireproofing is essential for this type of layout.

On smaller operations, buildings are usually of frame construction, well insulated, and segregated to conform with insurance underwriters' regulations.

Warehousing: Nonperishable goods are often stored in heated warehouses for the comfort of the issuing and checking clerks. Mill stores are usually kept in or adjacent to the mill to avoid outdoor handling. Even if transportation is on an uninterrupted basis, most of the incoming freight will be handled in the summer months. Thus inventories are higher and planning more difficult.

Power Generation

Cold weather hydroelectric generation presents difficulties because of ice formation at dam and tail races. Compressed air in the forebay has been used successfully to combat this.

Diesel power generation faces the following problems:

1) Very cold air drawn from outside for combustion means poor diesel efficiency.
2) Warm air drawn from inside the plant must be replaced by heated air.
3) Low pour point fuels are expensive.
4) Bunker *C* type of fuel oil will not flow at low temperatures and storage tanks, pipelines, and pumps must be heated.

Power transmission lines must be properly designed to allow for greater than usual expansion and contraction due to large temperature ranges.

Underground Shops

Some operations have been placed underground, on the theory that cost of excavation is not prohibitively high. In fact, it is comparable to buildings on a unit basis. Heating is supposed to be simpler and cheaper. However, ventilation must be provided, and almost invariably running or dripping water is a nuisance. Guniting and painting is usually resorted to and thus end costs may be slightly higher than for equivalent space on surface.

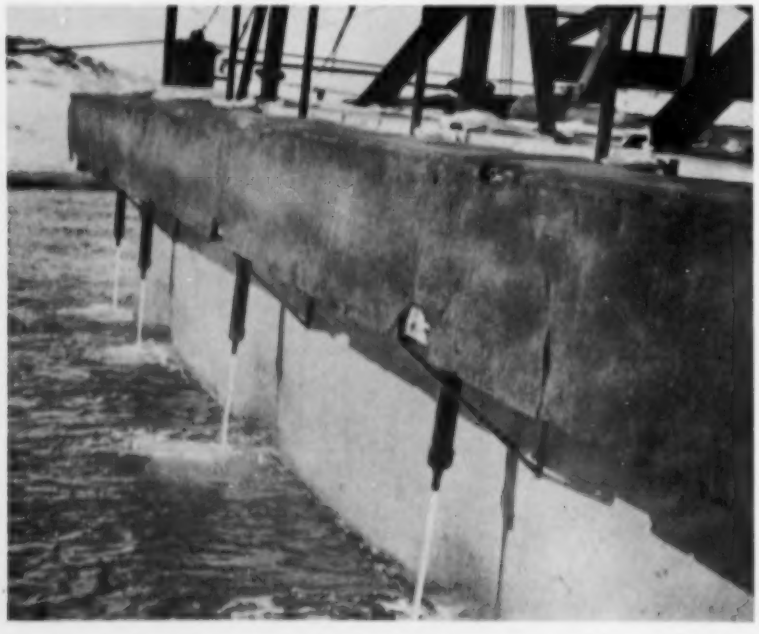
Head frames and machine shops have been placed underground with some promise of success.

Adit entrances to ore deposits are generally adopted for small mines or for development purposes. They are ideal for rapid progress and free from the cold weather problems of equipping a shaft site with collar and headframe. However, in nearly all cases, small operations that start with adits end up with internal head frames and eventually conventional shaft surface facilities for reasons of flexibility and better efficiencies at higher tonnages.

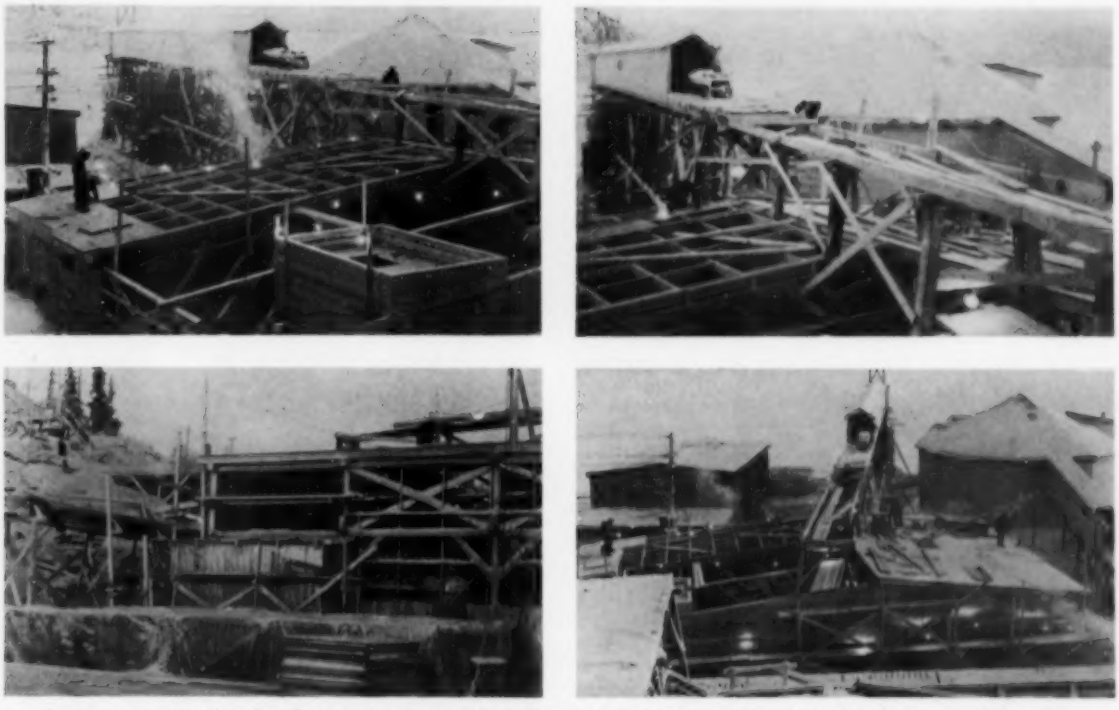
Conveyor ways from ore pass to mill site and from waste pass to disposal area can economically be placed underground. Use of upcast mine air for moderate heating is customary.

Mill

In the mill there must be vast storage facilities and heated space for all equipment items. Ore bins,



Side view of dredge showing nozzle action which keeps a channel open in the coldest weather. Average temperature of the water used in nozzles is —41°F mainly from underground. Total flow to nozzle is 780 gpm and the pressure at the nozzle is 50 psi. Note insulated box covering the manifold from which nozzles discharge.



Reconstruction of mill at Port Radium after fire completely destroyed the building in November 1951. Construction during the winter meant resumption of operations in the following spring.

thickeners, agitators, and pachuca tanks, which can be left outdoors in more southerly climates must be housed. An interesting pioneer move is currently being made in the area around Blind River, Ont. All thickeners in a counter current decantation circuit are being placed outdoors. Successful operation of this circuit may modify design in the future.

Heating of mill buildings can often be partially helped by process heat. Buildings are generally windowless. Waste heat boilers on diesel exhaust systems are common.

Mill construction, where process heat is not available, will normally be very elaborate to take advantage of the latest techniques in insulation and vapor sealing materials. Roof construction in particular is a problem. Three types of construction can be said to be reasonably successful and economic:

1) Cemesto panels with pitch-caulked joints and 20-year bonded surface.

2) Laminated wood deck at least 4 in. thick and often more, with 20-year bonded surface.

3) Double roofs. This system appears to be satisfactory and cheap. Typical frame construction would consist of a 2-in. wood deck with a mopped-on vapor seal—2 or 3 in. of insulation covered with a 1-in. wood deck nailed to spacers. A 20-year roof on top of this completes the sandwich and service has been excellent.

Many other types of roofs have been used, including Robertson deck with rigid fiberglass and 20-year bonded surface. Corrugated asbestos with a sprayed limpid asbestos interior is satisfactory if a vapor seal, such as plastic paint, can be satisfactorily applied to the asbestos fibers. Expansion and contraction must be considered in steel or rigid types of construction when large roof areas are involved.

As a general conclusion, it may be said that frame construction properly engineered is satisfactory and economical. With this type of construction, insurance costs are high and sprinkling installation is almost universal.

In this connection it should be mentioned that fire hydrants are costly. Heated and insulated houses for each hose station are essential. Steam tracing of the fire line is usual. Frequent inspection and actual tests are required to be sure that when needed a strategic hydrant will be free from frozen pipelines and valves. With stagnant water lines, this is a very real problem regardless of the amount of care taken in engineering or construction.

Tailings disposal lines in most cases must be flexible. Therefore, complete pipe box facilities are not usually installed. Flowing pulp very seldom freezes even in the coldest weather, so it is not unusual to use exposed iron pipe, properly graded so that in case of emergency shutdown the pulp will drain automatically. Wood stave lines have been used, but from cold weather point of view are not necessary.

Despite the best efforts, frozen tailings lines sometimes plague operations. When several thousand feet of pipe are involved, no simple solution to this problem has been proposed. It is common to have two parallel disposal systems for such emergencies, and in some cases a third is hastily installed when both lines are frozen solid. Electric heating of pipelines can be used for unblocking in case of local freezing.

As to the actual tailings pond, it can be the largest single headache in operations. The only known simple and cheap method of tailings disposal is to choose a lake that is deep enough to suffice for at least one full winter. Thus discharging tailings through a simple hole in the ice eliminates troubles for the winter. When this is not possible or practical, real troubles start. Tailings discharged from a pipe onto frozen ground freeze immediately and cone up very rapidly to the pipe discharge. Daily moves of the

pipeline are common. This problem becomes critical if reclamation of tailings is contemplated. Tailings ponds built up in this way are up to 90 pct ice and thus hard to reclaim.

Roads and Yards: In definite permafrost areas, road building outside the immediate plant area is sometimes very simple and cheap. As a road bed, permafrost is unequalled. In muskeg areas, it is merely necessary to place some crushed rock or gravel on top of the vegetation and the road is built. Settlement is impossible and maintenance is negligible. This is not true in areas of porous overburden, where surface water movement in summer may thaw ground to a depth of several feet.

Yards which are either man-made in the vicinity of the shaft or natural are essential for storage. Dumping of wet muck in very cold weather results in unstable yard area. Angle of repose may be as high as 80°. When spring comes, the material close to surface will thaw and slump, making track maintenance difficult. Usable yard space can of course be built in summer, and winter-built yard becomes stable after a year's settling. Even at that the interior of large waste dumps may remain frozen indefinitely. This makes such waste fill unsatisfactory as a foundation for buildings or heavy equipment.

Plastics and Natural Rubber: At very low temperatures most plastics and rubber goods become brittle and may be broken or damaged by normal handling.

Operations: Inside most plants it is unnecessary to consider cold weather as an operational problem, since the engineering and construction will have been properly carried out. However, heating costs are higher. As all inside operations are dependent on outside services, it may be well to consider these individually.

Transportation: Vehicles used for transporting men, materials, ore, and waste are particularly vulnerable to severe cold weather. It is safe to say that even with excellent maintenance and the most suitable lubricants, costs per ton mile are appreciably higher in the far north. If vehicles are not housed when idle, cold weather starting and operation will result in short engine life. Some vehicles are kept idling 24 hr a day, although the service for which they are being used requires only a few hours of operation.

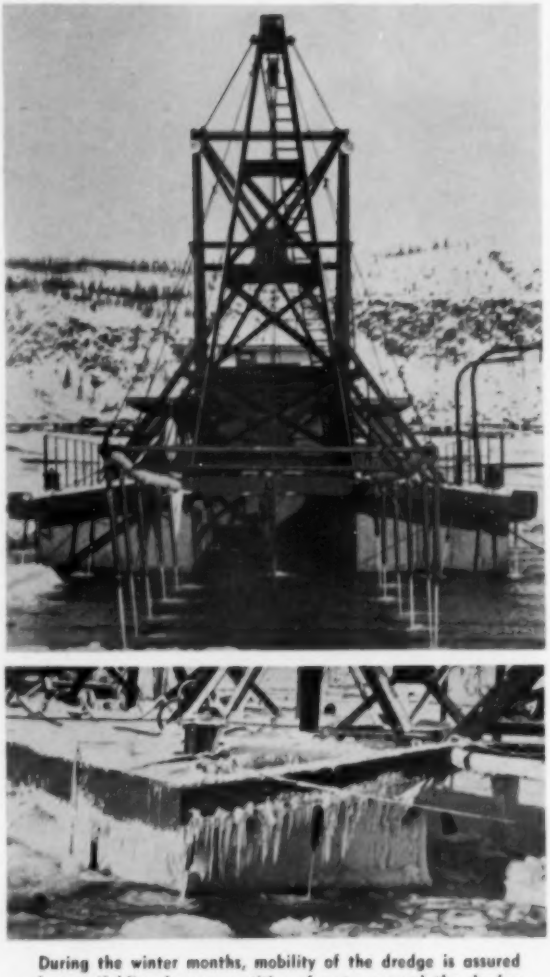
A variety of special lubricants have been developed for far north service. Special batteries are obtainable at high cost but are seldom used.

Spring and other failures are frequent and may be partially due to cold weather brittleness. However, as a rule, roads are rough and such troubles can be expected.

Tailings Disposal and Reclamation: This subject could have been discussed under Design and Construction but regardless of design, the operation can be very difficult. There are two cases to consider: 1) disposal and reclamation above water level, and 2) disposal and reclamation below water level.

In the first case, disposal problems are usually associated with the need for almost daily moving of the discharge end of the pipeline and the maintenance of proper drainage slopes in case of a pump failure.

Reclamation of such tailings can be economically carried out only in the summer months. Even then, ordinary methods must be modified. Dragline operations can be carried out to the extent that thawing occurs. A reasonably cheap operation of this kind would entail daily moves of the tail block so that a fan shape area is worked. By the time several cuts have been taken, the tailings under the original cut will have thawed to permit repetition of the operation. Water can be used to speed the thawing.



During the winter months, mobility of the dredge is assured by manifolding large quantities of water around the dredge hull.

Continuous sluicing of tailings to a central pump station is practical. Sluicing water acts as thawing and transfer agent.

As pointed out earlier, disposal of tailings through the ice of relatively deep lakes is cheap and simple. However, reclamation later is complicated.

Dredging[7, 8] of such tailings has been carried out successfully winter and summer. During the winter months, mobility of the dredge is assured by manifolding large quantities of water around the dredge hull. Ordinary lake water at or near the freezing point is used but is not as satisfactory as slightly warmer water. Water from underground and waste solutions or water from milling operations can be used to advantage.

Heating of the control house and maintenance of all dredging cables and pipelines are troublesome. A source of high pressure steam is advisable.

Underground Operations: Except for hoisting, caging, and decking, underground operations in the far north do not differ greatly from those in warmer areas. Properly designed facilities can eliminate nearly all operating difficulties.

Briefly some of the problems encountered are:

1) Rope dressing must be carefully applied to avoid excess that can build up on sheaves, freeze, and eventually cause the rope to jump out of the groove.

2) Sheave wheels and bearings must be cleaned and lubricated. Most sheaves are housed in to avoid this problem.

3) Ice in the head frame and sheave house builds up and condenses. This can be a major problem, since most shafts are upcast and humid air deposits water and ice on all interior cold surfaces or forms icicles at all ports. Proper engineering to prevent this is costly.

4) Nearly all operating mines in the north have large forced ventilation systems. This means heating all ventilating air for at least five months of the year. As a rule the quantity of air circulated during the winter months is reduced for economic reasons, but heating of 10,000 to 20,000 cfm is common.

5) Fresh unventilated headings are uncomfortable to work in and dormant air and water lines must be drained. Water from drilling operations freezes on the track and makes dangerous working conditions. Fortunately, after development is completed in an area, warm ventilation air causes the permafrost to recede into the walls and such conditions exist only for a short time.

Personnel: Some concessions in the way of higher wages and benefits are made to personnel in the far north, more because of isolation than cold weather. Long cold winters do, however, change the recreational habits of employes, and it is wise to take this into consideration when planning for northern operations. Normal outdoor activities such as skiing and skating are almost impossible. It is common to provide good indoor facilities to compensate for this. Curling, bowling, badminton, and volley ball can be readily provided. A good library and a theatre help to keep up morale. It is natural to expect high labor turnover, but here again isolation rather than cold weather may be the dominant factor.

Conclusion: Much more could be written on the subject and in particular a quantitative cost study would be of great value. A volume could be written on the engineering aspects of construction in permafrost areas. It is felt, however, that the present operations in the north are too few and as a rule too small to form the basis of such a study.

Experience gained to date indicates that operating costs need not be appreciably higher in the far north. Capital costs without any question will be much higher. In addition a greater responsibility falls on all the people connected with such ventures. In cold climates, engineering mistakes both before and during operations are much more costly.

Cost data from existing plants in the far north indicate that both operating and capital costs are more than double those of southern operations. In the case of capital costs, transportation charges make up most of the difference. However, a summary of other factors would include: 1) increased engineering costs due to novelty and complexity of design, 2) increased construction cost due to lower labor productivity in cold weather, and 3) increased material cost due to more elaborate structures.

From an operating point of view (and again neglecting transportation costs) the following factors would contribute to higher costs: 1) somewhat higher wages, salaries, and benefits; 2) increased heating load; and 3) higher costs of providing and maintaining services and recreational facilities.

Actual costs of similar milling facilities in northern and southern Canada have been in the ratio of 2:1.

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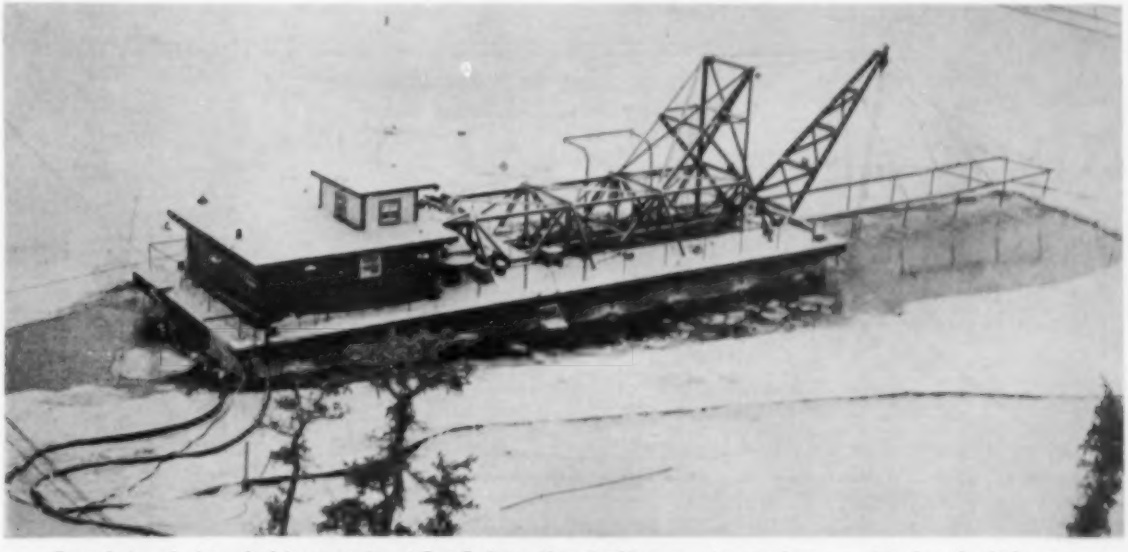

General view of winter dredging operation at Port Radium. Note flexible connections to shore—one 6-in. hose for high pressure water, to feed the nozzles surrounding the dredge, and one 4-in. hose to deliver dredged tailings to the processing plant. Also note deadman in ice to which cables are connected. A multidrum winch in the dredge makes maneuvering very simple and flexible. Power supply cable lies on ice near hoses.

Use of Belts and Concrete Ore Passes

by E. G. Williams

CONVEYORS were first used underground at Miami Copper Co., in the latter part of 1947. At that time some of the stoping area was too heavy and too close to the haulage level to use conventional haulage. To get the haulage drifts out from under the stoping area belt conveyors were used for the additional lateral transfer. Since 1947 more than 8 million tons have been mined in this manner, with as many as four belt type stopes in production at one time.

When the mining plan for the new low grade was worked out, it was decided to use belts on what otherwise would have been short train haulage transfers to move the ore laterally to centrally located concrete ore passes. One group of these stopes has grizzly levels on the 570, with a belt level on the 670; the other group has grizzly levels on the 720 and belt levels on the 820, with the concrete ore passes bottoming on the 1000, or the existing train haulage level.

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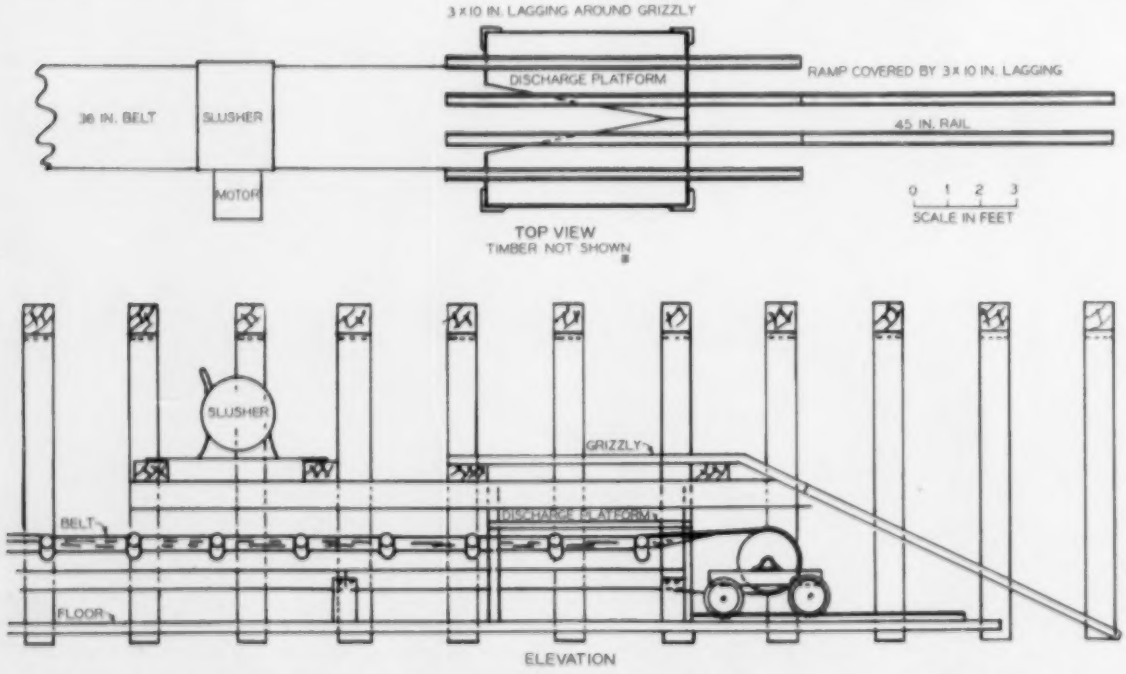
To handle the necessary tonnage, about 2000 tons a shift, a 36-in. belt was installed, driven by a 20-hp motor at 300 fpm, with capacity of 700 tph.

On the belt level a standard haulage drift, with 9-ft posts and 8-ft caps, allows plenty of room for passage, clean-up, and repairs. Two chutes in each of the pony sets are controlled by arc doors. For driving the drift a 25-hp slusher with a 42-in. scraper was used, 240 ft were driven, pony sets were installed, and 15 ft of each transfer raise were driven. The 15 ft of raise are driven so there would be no direct blasting on the belt. Then 200 ft of conveyor belt was installed, a ramp installed at the end, and drifting continued by slushing onto the belt. The belt could be reversed to handle timber for the drift and cribbing for the raises. An additional 180 ft of drift was driven in this manner, pony sets put in and raises started, then complete belt installed, this finished installation requiring a 400-ft belt.

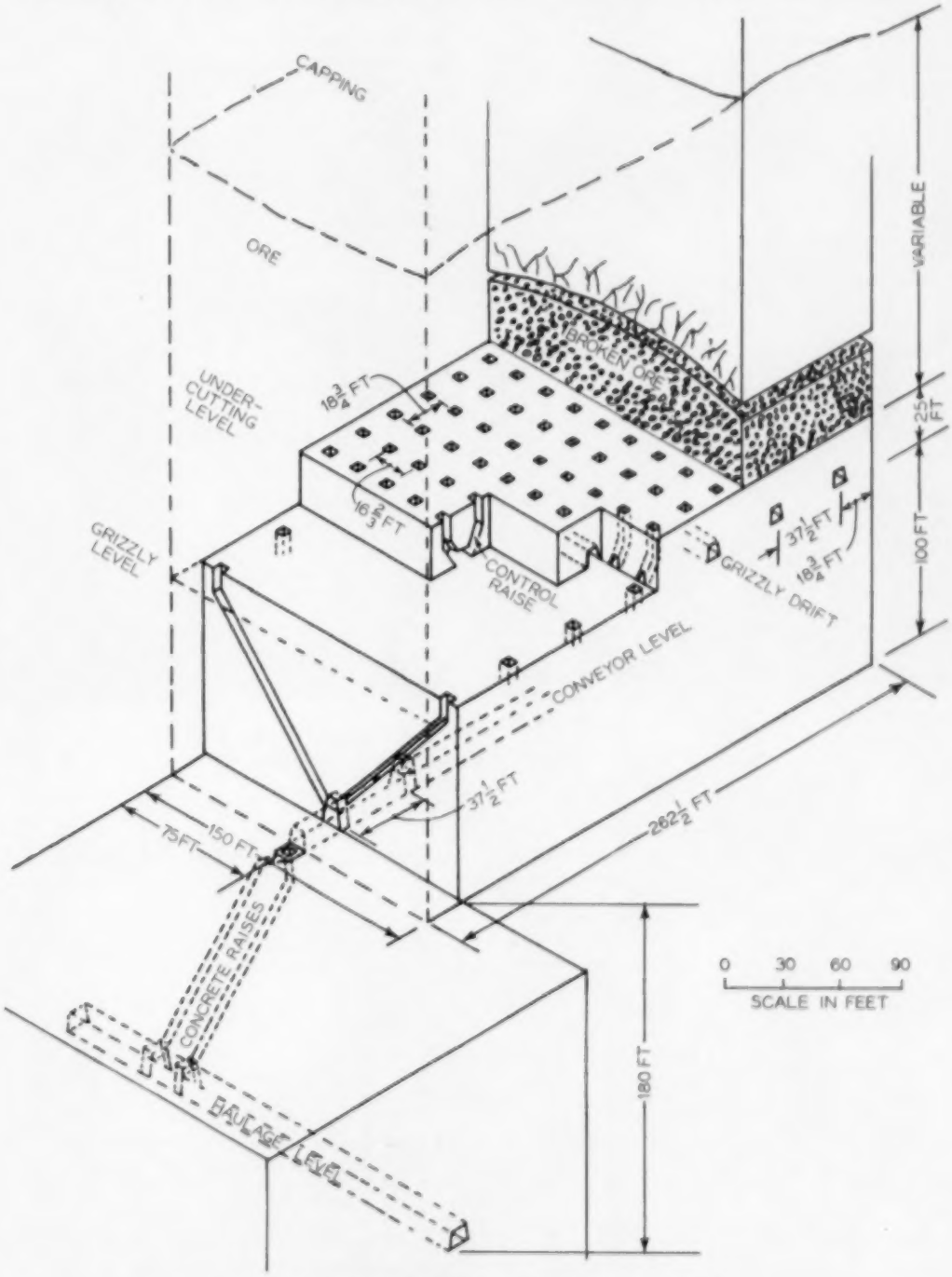
The next consideration was feeding the ore from the chutes onto the belt. This was done with a locally designed auxiliary chute or hopper. The chute should give the ore a velocity and a direction at the point



To handle the necessary tonnage, about 2000 tons a shift, this 36-in. conveyor belt was installed, driven by a 20-hp motor at 300 fpm, with capacity of 700 tph.

Miami Copper Co. Haulage System

Transferring ore from the 570 grizzly level to the 1000 train level by this system shows increase in production and better operating results.



Isometric drawing of typical caving block. At present about 60 pct of the tonnage mined by Miami Copper Co., Miami, Ariz., is stope mined by this method.

of contact with the belt equal to the velocity and direction of the belt itself. It should resist abrasion, be self cleaning, and center the feed on the belt. The importance of reducing to a minimum the impingement of the feed at the loading point cannot be overstressed as this is the cause of most belt damage. Also the chutes must raise easily to allow ore to pass from the chutes behind. After considerable experimenting the chute decided on was made of ⅜-in. plate, 2 ft 2 in. wide and 2 ft 9⅝ in. deep with a tapered slot in the bottom 2 ft 7 in. long and tapering from 1 ft 8 in. to 4 in. It is hinge mounted on a 3 in. heavy duty pipe by two hooks ⅜ x 4 in. and while the raises are being driven the front end is suspended by chains from a trolley wheel on a ½ x 3-in. bar, lagged across the drift. This allows the chute to be moved from side to side so that cribbing can be hoisted. After raises are complete and arc doors installed, chutes are put in a permanent position and front end counterweighed.

The first belt installed has eleven loading positions controlling twenty-two 100-ft raises fed by slushers. The belt discharges into two 4 ft diam concrete raises. The tops of the raises are designed so that both can be used at the same time or either closed off for repairs. Control points are located midway between conveyor discharge and haulage level and openings at two other levels in case of hang-up. A standard pony set is used for drawing these raises, but with a steel chute of ⅜-in. plate and a guillotine-type air gate of 1-in. manganese steel powered by a heavy duty 8-in. Ladeen cylinder with a 30 in. stroke. This type of gate has been used at the skip loading pocket for a number of years. The chutes are so spaced that two cars can be loaded at the same time.

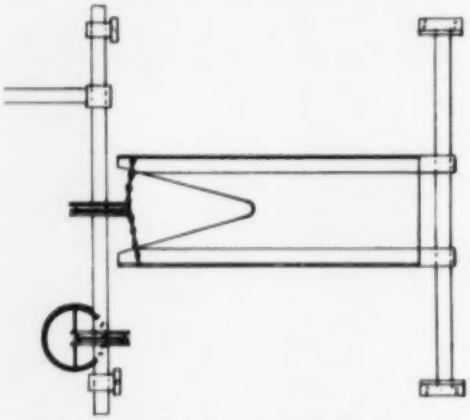
There are now three more belts in service, two on the same level as the first, but somewhat shorter (310 ft) having eight loading positions. These belts run 187.50 ft apart and discharge into double raises. The raises are then drawn on a belt running at right angle 150 ft below and conveyed to a concrete ore pass. While in the same position, this belt is used to pull another stope 100 ft above.

Operating results on this 36-in. belt—concrete ore pass system to date show some 1,984,000 tons drawn from the top 570 level stope since January 1955, the high month showing a draw of 220,000 tons or 7857 tons for a 2 shift day.

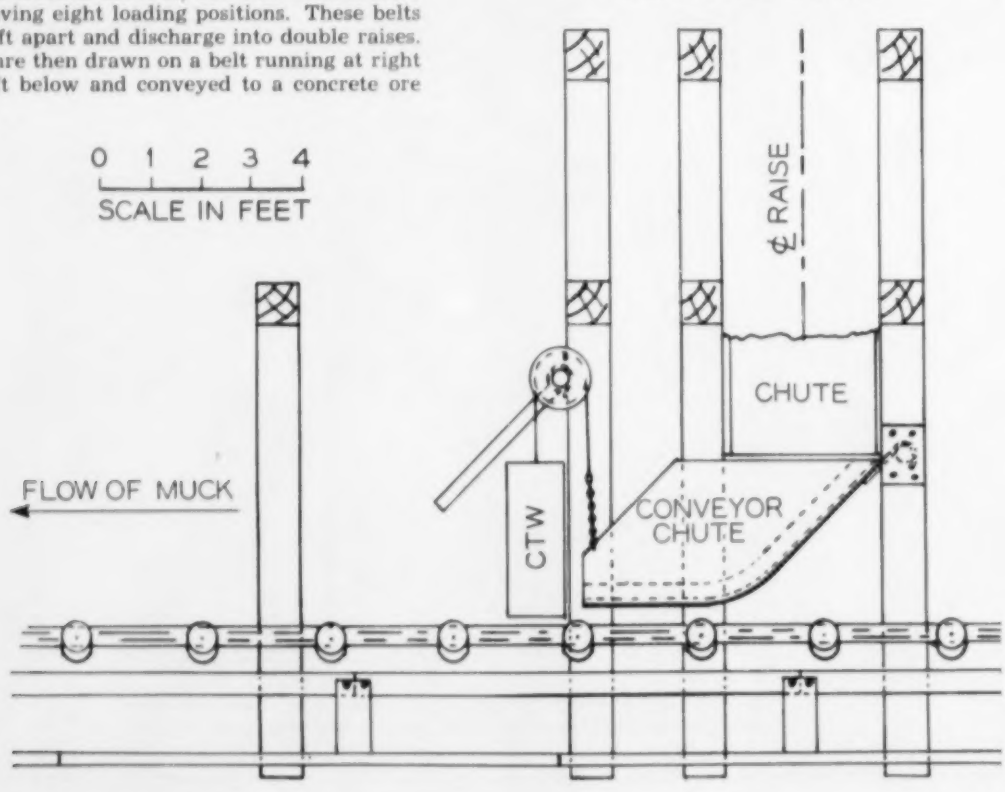
The third stope recently undercut shows a draw 262,000 tons to date, the current production from this type of stoping averaging a little better than 7000 tpd, the maximum expected per day being about 9000 tons.

Acknowledgment

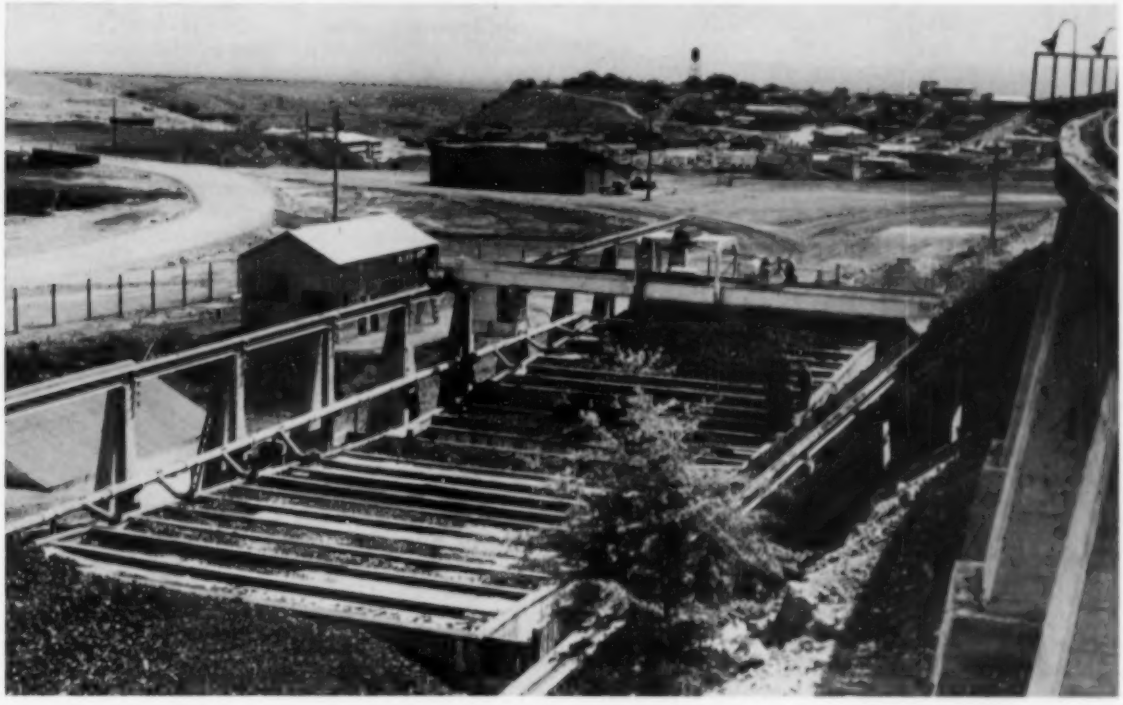
The author wishes to thank J. W. Still and J. B. Fletcher for their help in preparing this paper.



TOP VIEW OF CONVEYOR CHUTE
TIMBER NOT SHOWN



Cananea's Program for Leaching In Place



by Robert C. Weed

LEACHING in place at Cananea began in the 1920's on a limited scale. The first plants were small wooden boxes located underground in the Capote and Oversight mines, and output was low. Scrap iron was used as precipitant. Later several other small plants were started up. Leaching was limited entirely to underground areas, and production remained low.

The Veta precipitating plant, the first on a larger scale, was put into operation in the late 1920's. After a period of intermittent service it was rebuilt in 1946 and has since been running continuously.

In 1943 work on the Colorada open pit was started. This produced 40 million tons of waste, which averaged 0.20 pct copper. When these dumps had aged a few years it was decided to leach them on a large scale, and after considerable research and investigation a leaching system of pumps and pipelines was laid out and installed. At the same time the Ronquillo precipitation plant was built to treat the copper solutions from the pit dumps. This plant started operation in March 1953.

Leaching areas consist of the Colorada pit dumps and various mined out and caved underground stopes. The principal mineral, chalcocite, leaches readily in a weak sulfuric acid solution containing ferric iron. Enough pyrite is contained in the areas to produce an acid-ferric iron solution strong enough to dissolve the copper.

Each area is studied to determine the best method of distributing the water over it and the best method of collecting the copper-bearing solutions after leaching has taken place. Water is generally distributed by means of ponds, ditches, and sprays. Spraying has proved the best method of spreading the water evenly; however, in a dry climate some water is lost through evaporation.

After passing through the leachable material the solution is collected underground by dams and bulkheads and on surface by dams. An effort is made to give the maximum storage capacity practicable.

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TP 4251B. Manuscript, Jan. 10, 1956. New York Meeting, February 1956.

The Transactions papers appearing in Mining Engineering during 1955 will subsequently appear in AIME Transactions Volume 205 and may be permanently referenced to that volume.

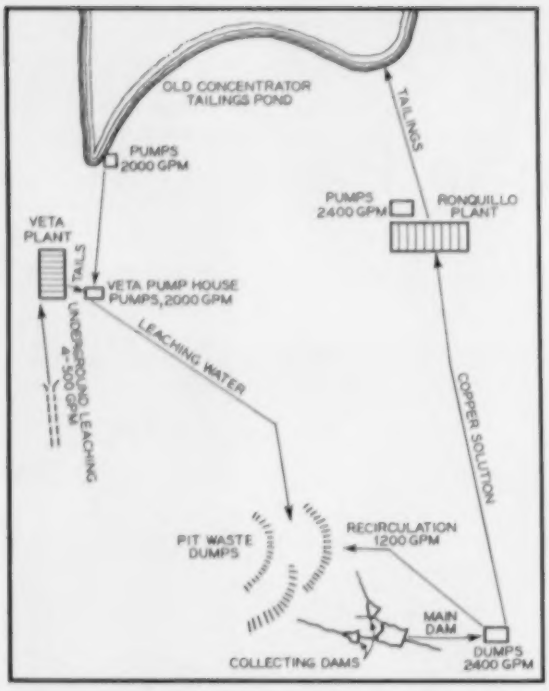

Fig. 1—The surface leaching system at Cananea.

For relatively low pressure (below 150 psi) transite pipe is used. For higher pressures iron pipe lined with wood or rubber is necessary. To facilitate repairs and replacements, pipelines are installed along roads or trails. Each line is equipped with a Y or a lateral to permit the insertion of a *go-devil* for cleaning. Periodic cleanings are necessary to eliminate build-up of iron ochre inside the pipes.

Pumps are mostly of the horizontal centrifugal type, installed wherever possible with a positive feed. Gages on pump discharge lines indicate the head the pump is working against, which can be used to calculate the pump capacity, and also show when the pipelines need cleaning. Nearly all pumps are equipped with automatic start-stop controls. Stainless steel equipment is necessary throughout.

Underground Leaching and Veta Plant: Water for underground leaching is obtained from present mining and from drainage of old mines. Roughly 400 gpm are used. Most of this water is pumped to surface, where it is sprayed over the top of the Colorada cave. From there it percolates down through the mined out sections of the Colorada mine and is collected behind a bulkhead on the Colorada 1300 level. It is then pumped to the 500 level tunnel, through which it flows to the Veta plant. Smaller amounts of water are being used to leach limited areas in the Veta, Oversight, and Republica mines. This solution also goes to the Veta plant.

As mentioned before, the Veta precipitating plant was rebuilt in 1946. Because there were shortages of cement and reinforcing iron at that time, it was decided to make the cells of wood. The plant now consists of 64 cells 4½ ft wide by 10 ft long by 5 ft high and 3 cells 9 ft wide by 16 ft long by 5 ft high. (The latter are for use of scrap iron as a precipitant.) Wooden grids with openings ½x¾ in. are placed horizontally in the cells 3 ft below the top to form a rack for the cans.

After the cell has been charged with burned, shredded cans, the solution is introduced and is directed up and down through the layer of cans by means of baffles. The water usually passes through a pair of cells. When the bulk of the can charge has been consumed, the cell is drained and the copper is washed through the grid with high pressure water. It flows as a sludge out of spigots near the bottom of the cell and down a launder into three large settling tanks with inclined bottoms. The bulk of the water is decanted from the first tank to the second, etc., and the cement copper mud is scraped up the incline with a 54-in. hoe-type slusher bucket. It is then loaded into trucks to be hauled to the concentrator and sluiced into the concentrate thickeners. Cans for the Veta plant are hauled from railroad bins by truck and stockpiled at the plant area. Later they are loaded with a rocker shovel into side dump tram cars, which are used in charging the cells.

Surface Leaching and Ronquillo Plant: Water for leaching the pit dumps is obtained from the old concentrator tailings pond, which collects tailings water from the Ronquillo precipitation plant and a considerable amount of water from the Democrata drainage area. This pond has a surface area of about 75 acres and an average depth of 60 ft. Because the water is acid and contains some ferric iron it makes a good leaching agent. Water is pumped from this pond to the Veta pump house at the rate of 1600 gpm. At this point it is joined by the Veta plant tailings, and the mixture is pumped directly to the Colorada pit dumps where it is distributed by means of ponds. Below the dumps the water is impounded in one of two collecting dams across the mouths of two deep gulches underlying the dumps. If the grade of the solution is too low, it is pumped to the top for a second pass. This solution is caught in the other collecting dam, drained to the main dam, and pumped to the Ronquillo plant, as shown in Fig. 1. After passing through the plant the tailings water is pumped back to the old concentrator tailings pond, thus closing the circuit. Excess iron precipitates out in the pond, probably as basic ferric salts, and the total iron content in leaching water remains about the same.

Table I presents a typical analysis of water in various parts of the circuit.

Table I. Analysis of Water in Circuit

Place	Cu, Gpl	Fe++, Gpl	Fe+++, Gpl	pH
Old concentrator pond	0.40	9.25	2.00	2.7
Veta plant heads	3.10	1.25	2.10	2.5
Veta plant tails	0.20	6.25	1.00	2.9
On dumps	0.35	7.10	1.80	2.7
Ronquillo heads	3.30	3.00	7.40	2.3
Ronquillo tails	0.25	14.00	1.50	2.9

The Ronquillo plant was constructed in 1952 after a study of a number of operating precipitation plants had been made. Influencing factors in the basic design were flexibility of operation, ease of materials handling, and availability of two overhead cranes, which could be adapted for charging cans into the cells.

The head water passes through a weir box with a recording meter and is distributed to the cells through a 12-in. pipe. This pipe has a number of takeoffs, each equipped with a valve, so that the solution can be controlled both as to location and volume of imput to the cells.

There are two sections containing ten cell pairs each. These sections are high on the outside ends and slope toward each other, meeting in the center of the plant. Solution can be introduced into the upper cell and can be taken out as tailings from the lower cell of any pair. This arrangement gives the plant flexibility and makes it possible to adjust for variations in volume and grade of head water. Usual practice is to operate three or four cell pairs as a unit. When a pair of cells is removed for washing, the water is carried from the pair above to the pair below by means of a bypass pipe.

The cells themselves are 40 ft long by 5 ft wide and slope from 4 ft deep at the back end to 6 ft deep at the front. The bottom slopes from the sides toward a center gutter that runs lengthwise. Stainless steel woven wire screens of ⅝-in. mesh are placed 3 ft down from the top to form a platform for the cans.

As shown in Fig. 2, cells are operated in pairs. The solution enters the front end of the upper cell, runs the length of it, crosses over a weir-like depression in the separating wall to the back end of the lower cells, runs the length of the lower cell, and crosses at the front into the upper cell of the next pair.

Cans are dropped from bottom-dump gondolas on a railroad trestle above the plant onto an inclined wooden slide, which carries them into a storage area under a craneway, Fig. 3, straddling the plant. They are charged into the cells by magnets operated from two overhead cranes. The crane and magnet combination is also used to help clean accumulated debris out of the cells.

Copper, washed through the grids by high pressure water, then flows down the gutter in the center of the cell and into a launder, which carries it to a stainless steel revolving trommel with ⅛-in. holes. Bits of iron, rags, and rocks carrying occluded copper are removed and shipped to the smelter. This coarse product contains 10 to 12 pct copper. The screened copper flows into four settling tanks, each 10 ft wide by 40 ft deep. The head halves of the tanks have sloping bottoms and truck loading facilities, in case it is ever necessary to scrape the copper into trucks for direct shipment to the smelter. The purpose of these tanks is twofold: 1) to provide adequate storage for plant production and 2) to help

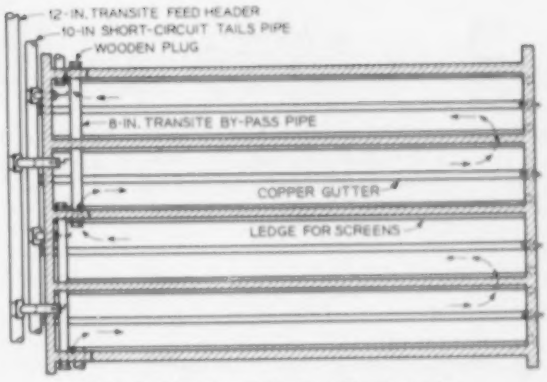

Fig. 2—Plan view of the Ronquillo plant cells.

remove, by decantation, as much iron in solution as possible. This soluble iron, if excessive, causes trouble in concentrate filtering. Overflow water from the tanks is returned to the plant cells, since a certain amount of copper is carried with it.

From the settling tanks the cement copper is washed into turbo-mixers that condition the pulp. It is then pumped as a slurry, varying from 5 to 15 pct solids, to a launder that carries it by gravity to the concentrate thickeners. Duplicate sets of two 4-in. rubber-lined pumps, each in series, are used to overcome a 170-ft total head. A velocity of 6 fps must be maintained in the 4-in. line to keep the copper in suspension. Lines are equipped with drains that are opened as soon as pumping stops. Solids in the slurry average 75 pct copper.

At the thickeners the cement copper from both plants mixes intimately with the concentrate. This prevents oxidation of the copper and reduces losses at the smelter, since the necessary sulfur is closely associated with the cement copper particles.

Conclusion: Open pit mining continues to make available large tonnages of copper-bearing waste, which is dumped in areas where is can later be leached. In mining underground (blasthole stoping) large pillars of lower grade material are left that slough eventually into the mined out stopes. Wherever it can be accomplished caving is induced to fill stopes with material from above. These caves often go to surface, which makes systematic washing possible and obtains some leaching from rain water.

During the past five years production of precipitate copper has gradually increased from 300,000 lb to as high as 1.6 million lb per month. Expansion of facilities is now taking place which should raise the output to 2 million lb per month. At the present rate of pit and underground mining, this output should be maintained for many years to come.

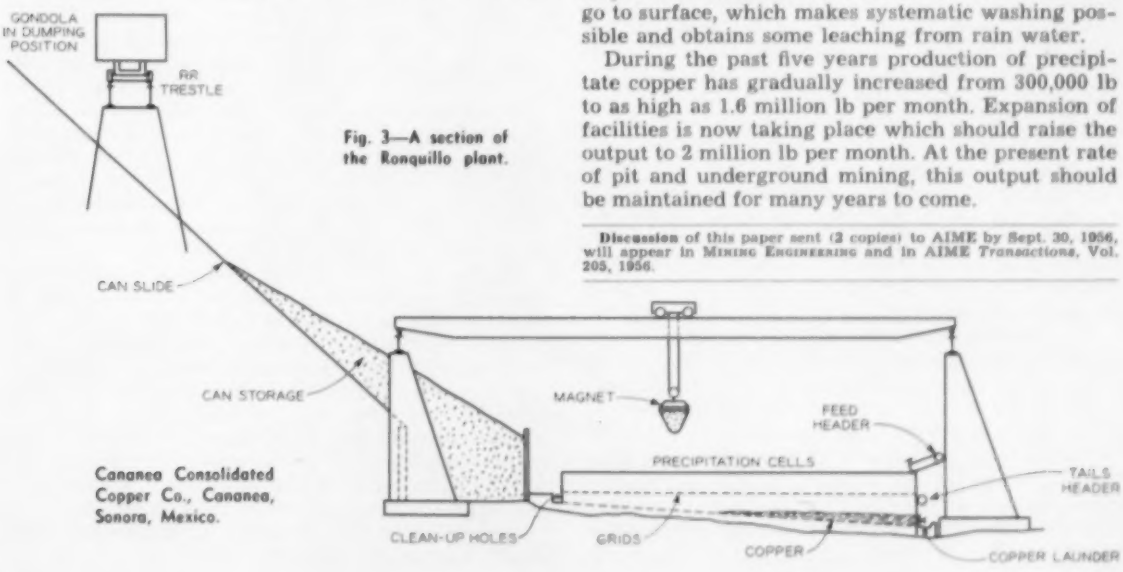

Fig. 3—A section of the Ronquillo plant.

Cananea Consolidated Copper Co., Cananea, Sonora, Mexico.

Discussion of this paper sent (2 copies) to AIME by Sept. 30, 1956, will appear in MINING ENGINEERING and in AIME *Transactions*, Vol. 205, 1956.

Raw Materials Preparation at The Brandon Plant, Mississippi

by J. C. Holm

ALTHOUGH the main constituents of Portland cement are the oxides of calcium, silicon, aluminum, and iron, characteristics of the cement are seriously affected by such contaminants in the raw materials as magnesium, phosphorus, sodium, potassium, sulfur, and others of lesser importance.

Sources of cement raw materials comprise a long list of minerals occurring in nature in various degrees of purity. The calcareous raw materials are generally limestone of purity varying from calcite to *cement rock* (a low-grade limestone of natural, almost correct chemical composition), chalk, marl, seashells, coral, and coral sand. The argillaceous raw materials supplying silicon, aluminum, and iron are mainly various forms of clay and shale, to which may be added silicious sand, iron ore, bauxite, or kaolin for minor adjustment of the chemical composition.

The quality of Portland cement in this country, as in most parts of the world, is controlled by rigid specifications limiting both its chemical and physical characteristics, and cement manufacturers must comply with these specifications to market their products.

Brandon Raw Materials: An unusual deposit of marl, limestone, and clay occurs 12 miles east of Jackson, Miss., near the village of Brandon. The deposit is of sedimentary origin and of relatively young geological age. Layers of marl vary from putty-like consistency to such hardness that it can be cut with a knife. Interspersed between the layers of marl are bands of limestone varying from very soft to hard, the compressive strength of the hardest layers exceeding 15,000 lb per sq in. The layers of marl range in thickness from a few inches to about 25 ft. The limestone bands seldom exceed 2 to 3 ft but often occur in thinner veins in the quarry face.

Above the top layer of the deposit, which may be marl or limestone, a layer of clay ranges in thickness from a few inches to more than 20 ft. Part of this clay is harvested with the raw material to supply part of the argillaceous components, but most of it is stripped as overburden and wasted. A highly silicious clay of extreme fineness, harvested from the bottom of the quarry below the marl and limestone bands, is to correct the silica content.

Fig. 1 shows typical examples of the interrelation of the various materials, based on drill logs obtained during exploration of the deposit.

The marls vary in moisture content in the bank from 15 to 30 pct and contain up to 70 pct −200 mesh particles and up to 40 pct −10 μ particles. A large percentage of the fine fraction is within the colloidal range, making the marl extremely sticky.

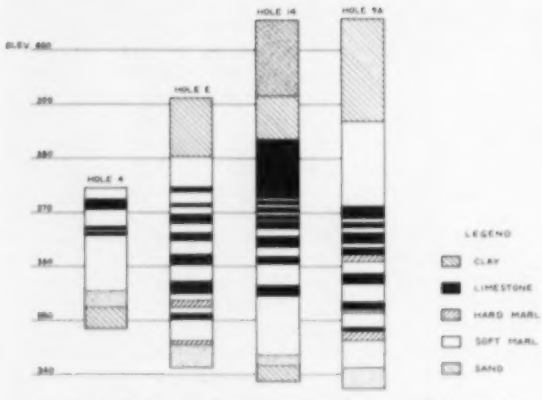

Fig. 1—Interrelation of various materials is shown by typical drill logs.

The limestone beds range in hardness from that of the hardest marl to a medium hard limestone and in moisture content from 2 pct to approximately 14 pct. The line of demarcation is often difficult to draw and the moisture content is possibly the best means of classification. The ratio of soft to hard material is not constant throughout the deposit, but the average is approximately 70 pct soft to 30 pct hard material.

Through the years this deposit had been repeatedly explored by cement companies and rejected as unsuitable for cement manufacture, mainly because of physical characteristics. In 1950, after careful exploration of the deposit and comprehensive laboratory and pilot plant tests of the raw materials, officials of Marquette Cement Mfg. Co. of Chicago were convinced that a practical and economical method of processing the raw materials had been devised and authorized construction of a new plant that began production in December 1951.

Marl and limestone, common raw materials for Portland cement, usually occur in such a manner that they can be worked separately. Their harvesting and processing normally have presented no difficult problems, as standard equipment and methods are available for handling either sticky or hard materials. No other cement plant utilizes a deposit of the nature of that at Brandon, however, and for this reason no prior art was available for design.

When properly blended, the raw materials produced an excellent cement without additives.

Basic problems to be solved in designing the cement plant were:

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TP 4241H. Manuscript, Feb. 10, 1955. Chicago Meeting, February 1955.

1) To design a primary crusher that would handle the mixture of sticky marl and harder limestone without choking.

2) To develop a layout and the necessary machinery to separate the hard, nonwashable raw materials from the washable soft marls and clay, so that the hard portion could be further reduced to mill feed size on conventional machinery, and to homogenize the softer parts, which vary considerably in chemical composition.

3) To be able to wash and separate, Fig. 2, with the minimum amount of water, as excess water would have to be evaporated in the kiln at the expense of fuel. It was evident beforehand that the water content of the kiln slurry would be higher than normally encountered in the cement industry, due to the large percentage of colloidal particles present in the marl.

After considerable research and test work, the flowsheet, Fig. 3, was developed and a new type of primary crusher and a continuous wash mill of radical new design evolved.

Flowsheet: The deposit is drilled with a Joy Mfg. Co. auger drill with air removal of the drill product. Holes are charged lightly, using decking through the soft bands. Broken material is loaded into 12-yd dump trucks, with side discharge using one 2-cu yd and one 3-cu yd shovel; hauled to the crushing plant; and dumped into the feed hopper for the primary crusher. The crusher is fed by a 72-in. heavy duty pan feeder.

Preliminary tests with a single roll crusher of standard design indicated that this type of crusher with some radical new features would handle the material. A 30x70-in. single roll crusher with movable breaker plate was developed, the links of which are equipped with teeth engaging between the teeth of the roll, as shown in Fig. 4.

Preliminary tests had proved that large slabs of hard marl or of limestone, which occur unavoidably in economical harvesting of the deposit, would merely come to rest on top of the roll and thus completely stop the feed to the crusher. The purpose of the movable breaker plate with its teeth is to tip these slabs on end and bring them down into the crushing chamber. By giving the teeth of the breaker plate a linear speed different from the peripheral speed of the teeth of the roll, it is possible to keep both roll and breaker plate clean even when very sticky material is being crushed.

The crusher, powered by a 200-hp motor, is easily producing its rated capacity of 300 tph when set to produce a −8-in. product. The principle of the crusher was developed by Marquette Cement Mfg. Co. and the crusher was designed and built by the Pennsylvania Crusher Co.

The crusher product is conveyed to the feed bin ahead of the wash mill by an inclined 42-in. belt conveyor. To prevent the sticky material from building up in the hopper under the crusher, one live smooth roll is installed on each side of the crusher discharge opening with the shafts parallel to the belt. Rolls are power driven and rotate in opposite directions towards the center of the belt, thus folding the crusher discharge onto the center of the belt. To keep them clean, they are equipped with scrapers, which have proved very successful. In fact, the output of the crushing plant drops to almost half when the rolls are out of order, and two or three men are then engaged continuously in cleaning the discharge hopper in spite of the fact that the sides of the hopper are almost vertical. The belt conveyor is equipped with water sprays and belt scrapers at the head pulley.

Washing and Separation: The feed bin ahead of the wash mill is designed with vertical ends and very steep sides on which fine water sprays play. A pan feeder extracts the material from the bottom of the hopper and discharges it into the feed spout of the wash mill. A dripple hopper with a ribbon conveyor at the bottom collects all sticky material that falls from the return strand of the pan conveyor and feeds it to the wash mill.

The horizontal continuous wash mill, consisting of a horizontal rotating cylinder equipped with lift-

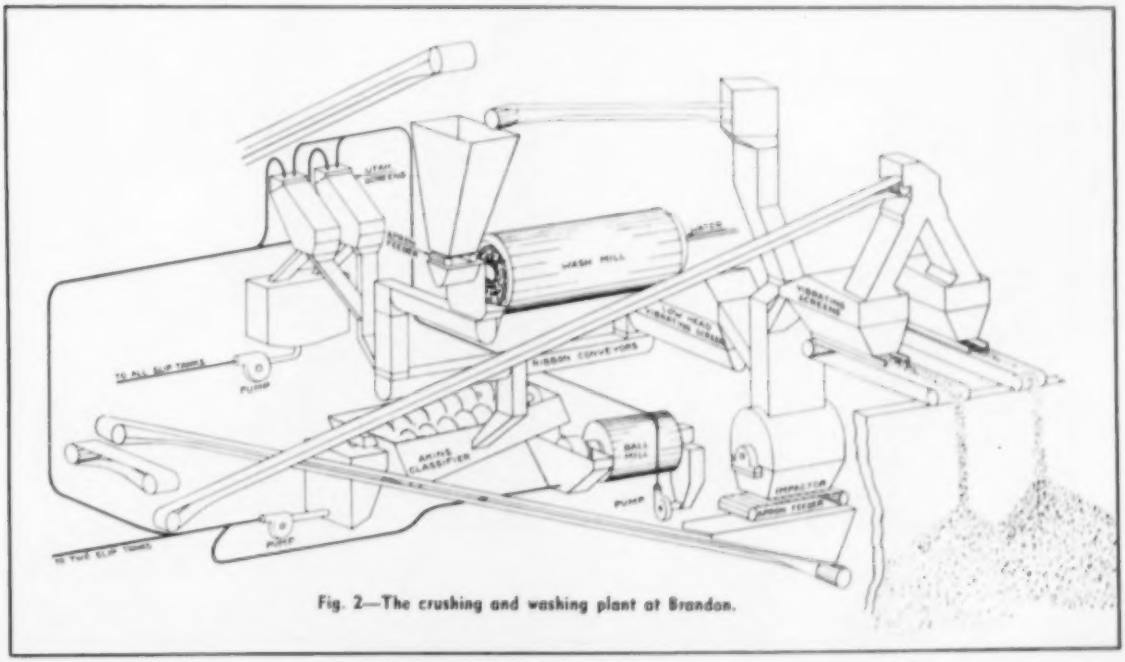

Fig. 2—The crushing and washing plant at Brandon.

Index of Plant Flowsheet, Facing Page

1	Raw material hopper		
2	Apron feeder	72 in.	Pioneer
3	Single roll crusher with traveling breaker plate	30x72 in.	Pennsylvania
4	Belt conveyor	42 in.	
5	Feed hopper		
6	Apron feeder	42 in.	Link Belt
7	Wash mill	11 ft 6 in. x 26 ft	F. L. Smidth
8	Low head vibrating screen	6x16 ft	Allis-Chalmers
9	Ribbon conveyors	36 in.	Link Belt
10	Impactor	ef 13-44	Pennsylvania
11	Apron feeder	48 in.	Pioneer
12	Belt conveyor	36 in.	
13	Vibrating screens (two)	5x8 ft	Symons
14	Belt conveyors (two)	30 in.	
15	Swivel conveyor	36 in.	
16	Classifier	48 in. ddps	Akins
17	Pump	8 in.	Morris
18	Ball mill	8x10 ft	Allis-Chalmers
19	Pump	8 in.	Morris
20	Utah screens (two)	4x6 ft	Allis-Chalmers
21	Pump	8 in.	Morris
22	Slip tanks with air agitation	55 ft diam x 24 ft	Dorr
23	Air compressors	c-110	Fuller
24	Mari pumps (two)	5 in.	Morris
25	Clay pump	3 in.	Morris
26	Mari ferris wheel feeder	36 in.	Traylor
27	Mari overflow pump	2 in.	Morris
28	Clay ferris wheel feeder	36 in.	Traylor
29	Clay overflow pump	2 in.	Morris
30	Limestone table feeder	8 ft diam	Falk
31	Clay table feeder	8 ft diam	Falk
32	Sand table feeder	8 ft diam	Falk
33	Belt conveyor	24 in.	
34	Compartment mill	8x29 ft	Allis-Chalmers
35	Rawmill pump (two)	5 in.	Morris
36	Blending tanks	22 ft diam	
37	Air compressor	c-110	Fuller
38	Air compressor	c-60	Fuller
39	Automatic timer for air agitation		Caine Sales Co.
40	Automatic sump level control valve	8 in.	Fuller
41	Sump		
42	Blending pump	8 in.	Morris
43	Kiln feed tanks with air agitation	55 ft diam x 24 ft	Dorr
44	Kiln feed pump (two)	5 in.	Morris
45	Slurry feeder		Allis-Chalmers
46	Kiln	12 ft 0 in. diam x 450 ft	Allis-Chalmers
47	Clinker cooler	744	Fuller
48	Clinker drag conveyor	18 in.	
49	Clinker bucket elevator		
50	Clinker distributing belt conveyor (with tripper)	24 in.	
51	Gypsum and sand track hopper		
52	Sliding plate feeder	30x72 in.	McLanahan & Stone
53	Gypsum and sand distributing belt conveyor (with tripper)	24 in.	
54	Bridge crane	8 ton	Harnischfeger
55	Limestone feedoweight	24 in.	Merrick Scale
56	Clinker feedoweight	24 in.	Merrick Scale
57	Gypsum feedoweight	24 in.	Merrick Scale
58	Drag conveyor	18 in.	
59	Ball mill	10 ft 6 in. x 15 ft	Allis-Chalmers
60	Bucket elevator		
61	Air separator	16 ft 0 in. diam	Sturtevant
62	Fringe tank		
63	Bucket elevator		
64	Tube mill	8x22 ft	Allis-Chalmers
65	Hummer vibrating screen	3 ft 0 in. x 5 ft 0 in.	Tyler
66	Cement pump	7 in. h2s	Fuller
67	Air compressor (for cement pump)	c-200	Fuller
68	Dust collectors (two)	8400 and 13300 cfm	Northern Blower
69	Cement storage silos (ten)	10,000 bbl each	
70	Portable cement pump (two)	8 in. h2	Fuller
71	Air compressor (two)	c-250	Fuller
72	Deaerator	5 ft 0 in. diam	
73	Deaerator dust collector	2100 cfm	Sly
74	Hummer vibrating screen	4 ft 0 in x 5 ft 0 in.	Tyler
75	Packer cement bin	2x350 bbl each	
76	Packing machine	4 tube	St. Regis
77	Packing machine dust collector	8000 cfm	Sly
78	Flexoveyor conveyor	32 in.	Flexoveyor
79	Spill screw conveyor	9 in.	
80	Spill elevator (bucket)		Link Belt
81	Bulk loading cement bin (two)	2x400 bbl each	

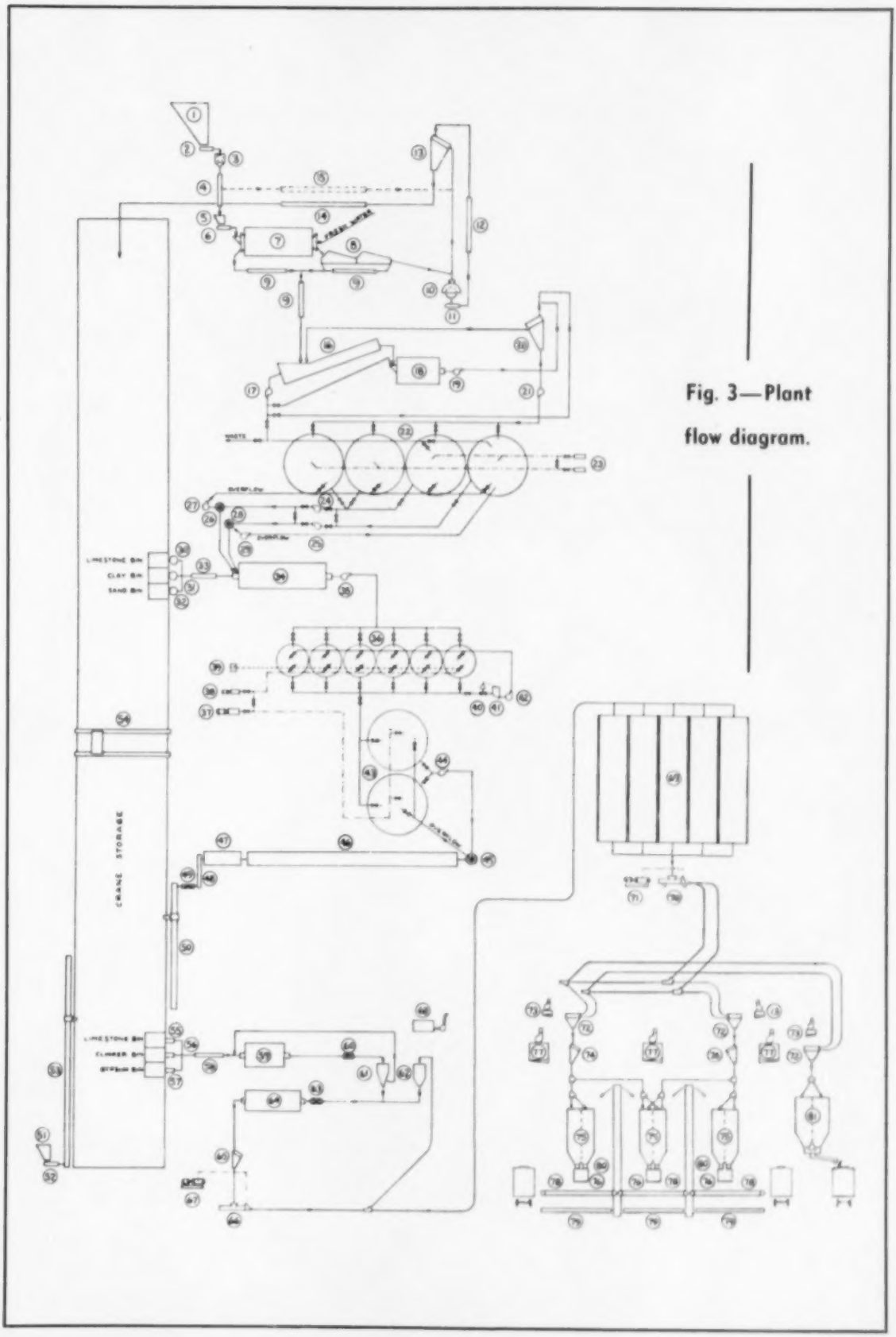

Fig. 3—Plant flow diagram.

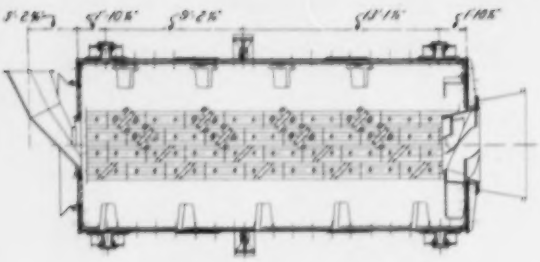

Fig. 4—Longitudinal section of wash mill.

ers or blades, has often been used, in the past, for separating sticky clay from rock and, in a cement plant in Sweden, for separating small amounts of flint from chalk. All these mills were designed as concurrent mills, operating in such a way that the raw material and water are added to the same end of the mill and the washable and nonwashable fractions discharged together at the opposite end. Actual separation and washing are done by vibrating screens or grizzlys with high pressure water sprays.

Clean limestone is necessary to obtain a chemically uniform product and to make it possible to crush the stone immediately after washing. The washable fraction of the material at Brandon at times amounts to nearly 75 pct of the total discharge. It was proved by tests that if a conventional wash mill were used under these conditions, a large volume of spray water would be needed, which would increase the water content of the kiln slurry excessively.

After considerable design work, the present writer conceived the counter current double discharge wash mill, Fig. 4, which differs from the standard mill in the following features:

The washable fraction is discharged from grates located around the feed inlet of the mill, while the nonwashable stone is propelled to the opposite end and scooped out of the center discharge opening. In addition, wash water is added to the mill at the end where the stone is discharged, forcing the hard fraction to pass in counter current through progressively cleaner water before being scooped out of the mill, while the water has to travel through the entire length to be discharged with the washable fraction. At times, when soft marl predominates in the feed, a small percentage of the water is added at the feed to prevent the ½-in. wide slots of the discharge screen from choking up.

Heavy manganese flights or lifters installed in the mill shell may be individually arranged to propel the material away from or towards the feed end. At the Brandon plant a pattern has been developed which gives the best average results.

The mill is 11½ ft diam by 26 ft long and driven at 26 rpm by a 600-hp synchronous motor through a speed reducer, which connects to a pinion engaging the master gear located at the center of the mill. The mill is supported on two slide shoe bearings, a type of support required for the design of the inlet end described above because standard trunnion bearings could not be used. This type of support also results in a vibrationless operation with extremely low maintenance cost.

A slide shoe support consists of a highly polished supporting ring of rectangular cross section bolted to the mill shell and resting on two shoes lined with babbitt, the surface of which is so formed that it does not damage the oil film on the supporting ring. One slide shoe support is fixed and equipped with a small horizontal slide shoe bearing located on each side of the ring to absorb horizontal thrust. The other support floats on steel rollers to allow for variations in the length of the mill shell.

The mill was constructed by F. L. Smidth & Co. of New York and has operated successfully since it was started. The wash mill is used in the installation as shown in Figs. 2 and 4.

The stone is discharged from the mill onto a 5x10-ft horizontal vibrating screen equipped with high pressure water spray nozzles, which are often not in use. The primary crusher is set at 8 in. to produce stones large enough to act as grinding media on the softer materials and to assist in bringing them into suspension with the water.

The stone drops directly from the screen into a Pennsylvania impactor from which a scissors belt elevates the crushed stone to two 4x6-ft Symons screens equipped with ½-in. openings and working in closed circuit with the crusher. The fines from the screen are discharged into the crane storage by two short high speed kicker belts.

It may be of interest to mention that the wash mill is producing stone so clean that a slight dust nuisance prevails around the crusher installation.

The washable fraction is discharged from the end grates around the mill inlet into a 30-in. ribbon conveyor and, together with the fines from the vibrating screen, is fed to a 48-in. duplex Aikens classifier. The classifier extracts the large quantity of broken limestone, contained in the raw material or produced in the wash mill, that is small enough to pass through the discharge screen. Underflow from the classifier is pumped to a 4x5-ft Utah vibrating screen equipped with 16-mesh screen.

Sand from the classifier is fed together with a fraction of the underflow to an 8-ft 6-in. by 10-ft 8-in., 300-hp ball mill, in closed circuit with another 4x5-ft Utah screen with 16-mesh screen cloth. Fines from both screens are discharged into a large tank from which the marl is pumped to the four homogenizing and storage basins. The fineness of the marl slip obtained by this entire operation is such that it all passes a 16-mesh screen with 60 to 65 pct passing 200 mesh and 50 to 55 pct passing 325 mesh. Moisture content is 55 to 65 pct.

The marl slip is pumped to four 55-in. diam circular slurry basins 20 ft deep, equipped with Dorr agitators, which homogenize the slurry efficiently, as no evidence of stratification has been found. Tank capacity is 3½ days full kiln output. Normally, one of the basins is reserved for high silica marl slip produced in the same way and harvested separately from the bottom of the quarry.

Control of Kiln Feed Slurry: Two conventional raw materials are produced—crushed limestone in the crane storage and marl slip in the tanks. Combining these two materials as feed to an 8x30-ft, two-compartment, 700-hp mill produces a slurry of 45 to 46 pct moisture. The slurry is pumped to six blending tanks from which the kiln feed is prepared, based on chemical analyses and using methods for proportioning that are well established in wet process cement plants.

The rest of the plant is essentially conventional in design and operating procedures.

Discussion of this paper sent (2 copies) to AIME before Sept. 30, 1956, will appear in MINING ENGINEERING and in AIME *Transactions*, Vol. 205, 1956.

Tailing Disposal At The Morenci Concentrator

by P. F. Allen

WITH capacity of 51,000 tpd, the Morenci concentrator produces approximately 49,000 tons of tailing for final deposition. Disposal involves distribution of thickened tailing to dams in such a manner that solids can be used for dam construction and water can be impounded for reclamation. During early operation, although tonnage treated was lower, it was necessary to use orthodox disposal practices in the areas available for tailing. Trestles mounted on a series of dams, constructed of borrow material, carried the pipelines from which tailing were spigoted behind the dams. When adequate disposal areas were finally established, it became desirable to look for more efficient methods.

The idea of placing the distribution pipe or manifold directly on the crest of the dam was advanced in 1945. After a period of testing, that procedure in modified form replaced the older trestle arrangement. Marked economies resulted, and time required to prepare dams for succeeding storage cycles was greatly reduced.

The tailing disposal area for the Morenci concentrator lies south and southwest of the concentrator building. Ample storage space is available. A wide valley contains three sharply defined ravines lying on an average slope of 4 pct. The more central of these ravines, and the largest of the three, is known as Morenci canyon, which provides substantially more tailing storage area than the other two. Stargo canyon lies to the west; the other is appropriately named East canyon. Total area of the tailing disposal site is nearly 3¼ sq miles, or about 2000 acres. The storage area in use was somewhat more than 850 acres at the end of 1954, or 16 acres per 1000 tons of tailing to be stored daily. At the present time none of the east side dams are in regular use but they are available in emergency. There are five dams on the west side (in Morenci and Stargo canyons) as well as two auxiliary dams adjacent to No. 4 West and No. 5 West. This has made it possible to concentrate men and equipment in a smaller area with consequent savings in both manpower and machinery.

Tailing Thickening: Tailing from the flotation process reaches the thickeners with a solid content of about 18 pct. The launder carrying this pulp is subdivided at a central point into seven launders carrying a proportional share to each of seven tailing thickeners, which are located in Morenci canyon close to the concentrator. Five of these thickeners are 300 ft diam and two are 325 ft diam. Thickeners are of the peripheral traction type and rakes travel at the rate of one revolution in 37 to 39 min. The entire installation provides 500,000 sq ft of settling area, or approximately 10½ sq ft per ton of tailing to be thickened daily.

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TP 4233B. Manuscript, March 22, 1956. New York Meeting, February 1956.

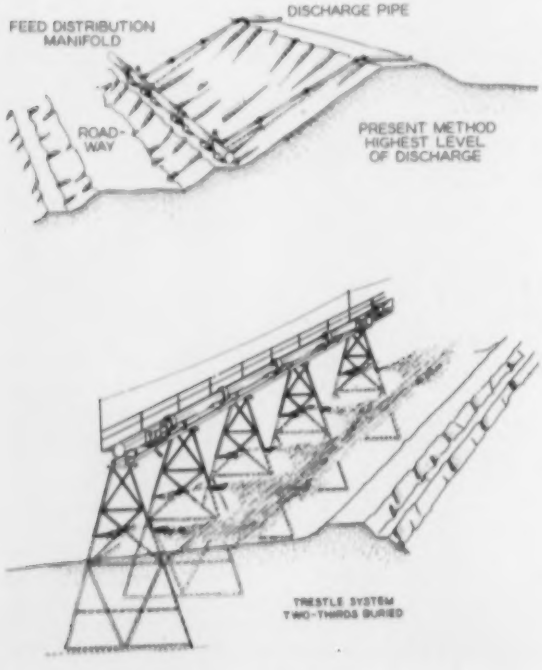
Sizing analysis of composite tailing for 1954:	
On 65 mesh	9.6 pct
100 mesh	13.6 pct
150 mesh	11.0 pct
200 mesh	9.2 pct
Through 200 mesh	56.6 pct
Composition of tailing:	
Pyrite	2.5
Quartz	39.0
Feldspars	23.0
Sericites	13.5
Clay substance*	22.0

*Including chlorite and ferruginous material.

The quartz and feldspars are the major components of the coarser screen sizes and the sericites and clay substances of the finer sizes, particularly the —200 mesh.

There are three 8-in. underflow pipes from each thickener. Through a labyrinth of tunnels the underflow pipes from five of these thickeners converge to a central discharge point known as No. 1 spigot house. The six pipes from the two larger thickeners discharge at another point called spigot house No. 2. Each discharge line contains two diaphragm valves. The discharge point of each pipe is equipped with a plug valve just ahead of an interchangeable spigot. These spigots, with uniform OD 7¾ in., have varying diameters for the orifice in the center. Used conveyor belting is fabricated to this purpose. Solid content of the discharged tailing varies from 45 to 55 pct. This solid content relates directly to the settling rate, which in turn depends on the type of ore and temperature of the pulp. Reclaimed water is returned to concentrator head tanks by two pump stations adjacent to the thickeners.

Transportation and Deposition of Tailing: Two tailing launders are arranged to receive thickener discharges at the spigot houses. One of these traverses the east side of Morenci canyon and the other skirts the west side of Stargo canyon, Fig. 1. The east side flume, 17,000 ft long, was a part of the original construction. The original launder lies on a minimum grade of 1.5 pct with maximum grades of 2.5 pct. Construction was of cement-asbestos composition. Launder shape and high gradients induced excessive wear, so that it ultimately became necessary to sheath it with concrete. The west side flume, laid on a gradient of 1 pct throughout its entire length, is of reinforced concrete.



Diversion gates and valves are located at points of convenience along the two flumes to facilitate necessary changes in routing the tailing. Laterals transporting the tailing to the dams from the flumes are either 24-in. pipe, or concrete launders measuring generally 36 in. wide by 24 in. deep. The launders, when used, feed into asbestos-cement pipe laterals at an elevation sufficient to provide a pressure head to carry the tailing through the horizontal laterals on the dams. Under prevailing conditions a minimum of 5 ft of head is required for every 1000 ft of feed manifold on the dam, plus 1 ft of head for every additional foot the tailing has to be elevated for discharge over the dam.

Construction and Operation of the Tailing Disposal System: In the first stage of dam building, distribution manifolds are placed on the crest of the dam. Made of cement-asbestos material, they are 24 in. ID. Manifold construction is with 10 to 13-ft sections, connected to each other by means of a collar of the same material and with round rubber gaskets to complete the seal. At intervals of about 52 ft, 6-in. spigots with plug valves are installed to discharge the tailing. The total assembly is calculated to withstand a working pressure of more than 100 psi. Berms are constructed with tailing in a series of 6-ft lifts by means of draglines, but the horizontal feed manifold is raised only once in every four lifts. This procedure means that for the intermediate lifts the only change needed to accommodate the change in elevation is to place an extension on each spigot pipe to carry it over the dam.

One operator and two helpers are generally required on each shift. When tailing is flowing into more than one dam simultaneously an additional operator may be needed; however, at the present state of development this is seldom necessary. Better than 3000 tpd of tailing will discharge from a single 6-in. spigot; therefore to dispose of nearly 50,000 tons of tailing about 15 spigots must be operated continuously. As these spigots are 52 ft apart, the care of 728 ft of bank is required. The operator and one helper attend to that task, while the other helper discharges all miscellaneous duties, such as checking on proper withdrawal of water through ducts or drainage stacks. The operator acts as lead man and is responsible for correct performance of the shift's work.

Dams vary in length from 0.3 of a mile at No. 1 West dam to 1.8 miles at the No. 5 West dam, see Table I. An access road traverses the top of each dam. Slightly to one side or on a berm at somewhat higher elevation rests the manifold, see cut. At intervals of 500 ft, drainage spigots connected to the feed manifold also permit disposal of tailing to the dam next below if required. Air relief outlets are provided at the ends of the feed manifold. Incandescent lighting placed along the manifold level of the dams and flood lights on the top of the berm facilitate operating the system during night shifts.

Water Reclamation: The water reclamation system at Morenci is so intimately related to the tailing disposal that it is necessary to describe in some detail the physical makeup and operation of the system. Beneath the dams there are three concrete ducts extending separately to the three canyons, two of these joining at the lower end of the area to deposit their combined reclaimed water into a single settling pond from which the water is withdrawn into a pump station. This is the No. 2 pump station, which receives all reclaimed water from Stargo and Morenci canyons. The East canyon duct ends at No. 3 pump station.

Table I. Data for Principal Dams

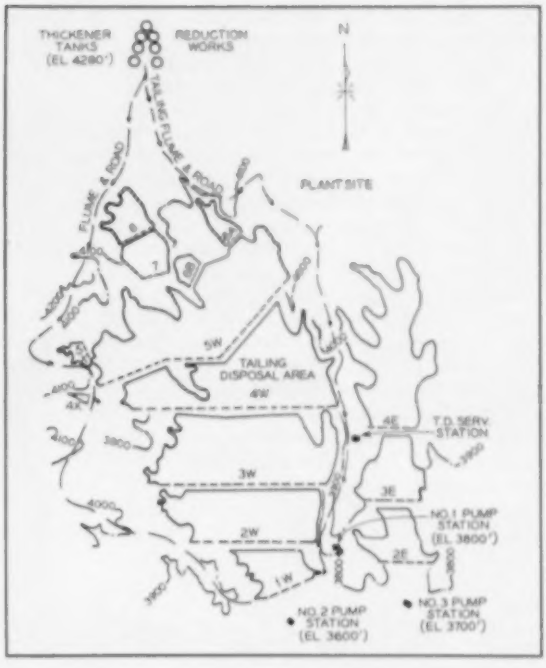
West Dams	Length, Miles	Acreage	Reclamation Stacks, No.
No. 1	0.3	44.71	None (Ducts used exclusively)
No. 2	0.6	137.98	2
No. 3	0.9	216.71	2
No. 4	1.3	212.98	1
No. 5	1.8	237.52	2

Sections of these water reclamation ducts were built as need developed early in the operation. Eventually all were connected until the main drainage system was complete. Construction of drainage ducts has by no means halted, since it is continually necessary to lay duct ahead of the encroachment of the back edge of the tailing settling area. These ducts are all laid on natural ground, following the original slope of the canyons. Eight-in. openings just above the floor of the duct decant the water until slimes encroach on the open hole, which is then stoppered with a wooden pug. At present the main drainage duct for Stargo canyon is 11,000 ft long; for Morenci canyon, 13,000 ft; and for East Canyon, 6000 ft—a continuous distance of more than 5½ miles.

Circular concrete stacks 6 ft ID, with walls 12 in. thick, were constructed wherever necessary. These stacks connected with the drainage network beneath the dams so that when the ducts were completely covered, which happens on most dams early in the operation, water could be decanted through holes of 8 in. diam arranged in a helix around the collecting stack. These holes are spaced nine to the turn and rise at the rate of 2 in. for each hole. As water is decanted and slimes reach the level of the open hole they are closed off with tapered wooden plugs.

In some cases water may collect in ponds remote from any reclamation ducts. When this occurs it may be recovered in two ways. If the pond is not too remote and is located against the bank of the next higher dam, the simple expedient of allowing tailing to backfill against the dam forces the trapped water out and over any obstructions to the stack. In the past, for some of the more remote areas, it was sometimes necessary to use portable pumping units mounted on skids and equipped with flexible suction hoses, the ends of which were provided with check valves and floated on empty oil drums. A lightweight rapid-connecting portable pipe was used to transport water to the nearest reclamation duct.

Safety dams are provided below the pump stations for protection against sudden influx of water. These constitute a temporary storage device, and the water is pumped back into the settling pond as soon as possible.



No. 1 pump station acts as a booster between gathering stations below the dams and the tailing thickener pump stations. It is equipped with nine two-stage 1000-gpm pumps, which return the water through appropriate pipelines to the pump station adjacent to the thickener installation. Static head on this station is 425 ft. No. 2 pump station, located below the principal settling pond at the junction of the Morenci and Stargo canyon ducts, is equipped with eight one-stage 1000-gpm pumps that elevate the water 305 ft to the No. 1 pump station. No. 3, which also pumps its water to the No. 1 station, elevates its pumpage 227 ft with four single-stage 1000-gpm pumps. Two men on each shift attend all pump stations.

Water reclaimed from the dams varies from 1 to 8 million gpd, averaging 5,582,000 gal per operating day during 1954. That amount is equivalent to 111 gal per dry ton of ore treated in the concentrator during the year.

Development of Present Disposal Method: At the beginning of operations, tailing was disposed of by the orthodox method of trestle-supported pipe systems, p. 726. High cost of trestle construction, total loss of material used, and operating difficulties involved made it desirable to seek new methods. The idea was ultimately advanced that the distribution pipe be placed on the crest of the dam so that the tailing could be discharged to the pond area by short lengths of spigot pipe. This eliminated the trestle but presented two possible disadvantages: 1) the hazard of dam washouts because of pipe failures or careless attendance, and 2) potential increase in the number of times the distribution line would have to be disassembled and reassembled. A test of the idea was initiated on a small dam. In a test period of two years, during which nine cycles of operation were completed, it was established that the hazard of washouts was unimportant. It was also found that with adequate head on the line, tailing could be discharged more than 25 ft above the feed manifold. Inasmuch as this testing had been

done on one of the very short dams it remained to be seen if this method would function adequately on great lengths of dam. While waiting to build a new trestle on one of the larger dams, it was decided, because of need for storage space, to utilize the new method along half the bank while awaiting completion of the other half. This test proved so successful that plans for building the trestle were abandoned and the pipe was laid on the crest for the entire distance across the dam. Subsequent usage proved the practicality of this method and resulted in complete conversion on all the dams.

Morenci System vs Trestle System: Prior to constructing trestles it was necessary to build a mat of dry tailing or earth fill across the dam to provide reasonable footing for the new construction. This meant trucking a substantial quantity of dry material. Even with such foundation construction, there existed the hazard of subsidence of the trestle before its full usefulness could be realized. Subsidence in some cases meant separation of the distribution pipe, with resulting spillage of large quantities of tailing that would wash out the dam. The present method of tailing deposition, while not entirely eliminating the possibilities of a break-through, does offer less opportunity for catastrophic incidents. It eliminates much of the clay near the face of the dam. It apparently classifies the tailing, depositing the sands and allowing the slimes to flow off. This is a strengthening factor, since there is now very little cracking of the surface upon drying, so that a better seal is maintained. Another factor of importance has been the shortening of time necessary to prepare a dam for re-use when movement of distribution manifolds becomes necessary. Most important, however, has been the substitution of the simple periodic movement of the manifold pipe for the more expensive trestle construction.

Discussion sent (2 copies) to AIME before Sept. 30, 1956, will appear in MINING ENGINEERING and in AIME *Transactions*, Vol. 205.

Mining Methods at the Iron King Mine

by H. F. Mills and L. Bombardieri

IRON KING mine, producing gold-silver-lead-zinc ore, is 10 miles east of Prescott, Ariz. At present the 1806 level is being developed. The echelon pattern of ore deposit continues at depth but is less pronounced, and the orebody is now more or less continuous over 2600 ft of strike length, with widths 5 ft to as high as 26 ft where overlapping of lenses occurs in one section of the mine. Better continuity of ore at depth has encouraged systematic mining and improved plant facilities to offset higher operating costs.

With favorable persistence of ore at depth, the three-compartment No. 6 hoisting shaft appeared inadequate to handle increased tonnage from greater depths, and in 1952, a four-compartment steel and concrete shaft was started. This is now at a depth of 1906 ft, and shaft pockets and loading cartridges are being installed on the 1806 level. Ore from the new shaft, hoisted in 5½-ton Jeto bottom dump skips, is dumped into a 500-ton shaft bin, from which it is fed to a 16x30-in. gyratory crusher and conveyed 415 ft through an undersurface concrete gallery to the secondary crusher and conveying system supplying the 900-ton mill. Lead, zinc, and pyrite concentrates are floated, and the flotation tailings are cyanided after desliming. Waste from the new shaft is dumped into a 250-ton surge bin, fed from this bin to a 24-in. conveyor, and discharged into the glory hole above the block cave. The new shaft is served by a double-drum 400-hp hoist used for ore and waste; the No. 6 shaft conveys men and material.

To secure a cheap source of waste fill for stoping operations, a block cave area was undercut in the

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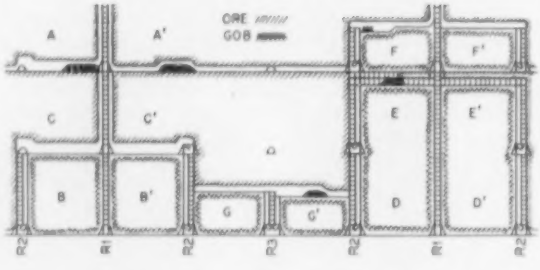
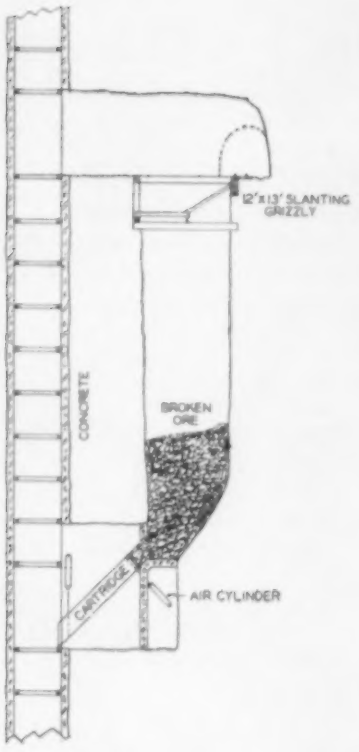
TP 4238A. Manuscript, Jan. 3, 1956. New York Meeting, February 1956.



hanging wall near the north end of the mine above the 600 level. Waste from the block cave is drawn from any of three large drawpoints on the 600 level, and a series of hanging wall transfer raises with controls and bypasses on each level provide a fast low-cost material for fill. The block cave is worked vertically through to surface in the form of a steep-sided hole 350x250 ft and several hundred feet deep, the sides of which cave and provide fill material without any blasting. Fill is a mixture of caliche-cemented surface gravel and small particles of hanging wall schist.

Level Development: An orebody 300 times as long as it is wide requires a lot of development work. Former practice consisted of driving a haulage drift about 30 ft from the ore in an andesitic footwall, a drift on the ore, and crosscuts from the haulage drift at 100-ft intervals. Crosscuts are about 90 ft long. At present, only the odd numbered levels are so developed, and on the even numbered levels only the haulage drift and crosscuts are driven. Little or no timber is used for this development. This change resulted in 20 pct reduction of development costs and 50 pct reduction of square set stoping required for removal of floor pillars. Elimination of ore drifts on alternate levels leaves



a solid block of ore extending from the back of the lower level to within about 12 ft of the ore drift above—240 ft as compared with the previous 115 ft.

Levels are planned for 133-ft intervals or for a total stope height of 266 ft. However, when a stope reaches the crosscut from the haulage level above, chutes are set in the crosscut and the stope is serviced from the upper level. It is seldom that ore passes are worn out before abandonment. During ore slushing the tops of the passes are protected by portable grizzlies. Little secondary blasting is needed and there is little trouble pulling ore from chutes.

Stope Development: All crosscuts and raises are on 100-ft coordinates, see cut, p. 728. At intervals of 400 ft a three-compartment raise (R1) with stulled and lagged center manway is driven on ore to the level above. The back of the ore drift is then taken out to 16 ft above track level for the 100-ft stope length (A) on each side of this raise. Track and pipes are removed, and two-compartment raise sets (R2) placed about 100 ft from the first raise. A gob floor is laid, and waste is dropped down the three-compartment raise and slushed out to fill the stope to within 6 ft of the back. A heavy plank is lashed to the slusher bucket and the surface of the fill is dressed smooth enough to permit laying the floor without using sills or spikes.

Stopes are mined in pairs. Steeply inclined stoper drills are used to drill and break ore in 6-ft cuts. Weak walls limit the depth of cut, and each slice must be filled as soon as the ore is slushed out. Before filling with waste, the 6-ft lengths of flooring are stood on end along the walls (F). The ore, while hard, is structurally weak, and as the stope advances it is usually necessary to square-set the floor pillar below the upper ore drift (E). When the stope has reached the level above, the block of ground remaining between the two-compartment raises is ready for extraction. The drift back is taken down and loaded out, and a three-compartment raise set (R3) placed at the midpoint, in the crosscuts. Manways of the two-compartment raises now become waste passes for the new stopes, and the new three-compartment raise, carried up with the stopes, is used for ore passes.

A cross section of the standard shaft pocket at Iron King mine is shown at left.

All stoping is on contract, see Table I, and is based on cubic feet of ore. The cycle is divided into four parts and weighted as follows: drilling and blasting 23 pct, slushing ore 43 pct, handling floors 12 pct, and waste filling 22 pct. All drilling is done with tungsten carbide bits and alloy steel rods. Early in 1948 shrinkage stoping was abandoned and replaced by cut and fill methods. In June 1948, after some time studies were made, the present contract system was established. No change of contract rates has been made since then, but contract earnings have increased by an amount proportionally greater than the miners' base rate of pay.

Table I. Contract Schedule, Iron King Mine

	Stoping, Cost Per Cubic Foot, ¢				
Average Width, Ft.	Floor, 12 Pct	Slushing, 43 Pct	Waste Fill, 22 Pct	Drill and Break, 23 Pct	Total, 100 Pct
12 or over	1.0	3.2	1.7	1.8	7.7
10 to 12	1.0	3.4	1.7	1.8	7.9
10	1.0	3.5	1.8	1.9	8.2
9	1.0	3.6	1.9	2.0	8.5
8	1.0	3.8	2.0	2.1	8.9
7	1.1	4.1	2.1	2.2	9.5
6	1.2	4.4	2.2	2.4	10.2
5	1.3	4.9	2.5	2.7	11.4
4 or under	1.5	5.6	2.8	3.0	12.9
	Taking Pillar Under Mat, Timbered Stope				
Any width stope, 5.0 timber	2.0	5.0	3.0	5.0	20.0

Square set timbered manway and chute, $6.00 per ft
Square set timbered manway and two chutes, $9.85 per ft
Single cribbed manway or chute, $3.85 per ft
Gob fence, $3.85 per ft
Base (drifts, $4.00 per ft over base price):
  Footwall, 2½ ft per round
  Ore, 3 ft per round
  Crosscuts, 3½ ft per round
Two-compartment raises (6x12 ft), $14.00 per ft
  $6.00 per ft of ground
  $8.00 per ft of timber
Three-compartment raises (6x15 ft), $15.00 per ft
  $6.50 per ft of ground
  $8.50 per ft of timber
Raw raises (6x6 ft), no staging, $3.65 per ft
  With staging, $4.85 per ft

This system of mining requires a large amount of stope preparation in advance of extraction, but it permits good control of grade of ore, and it has been possible to compensate, to some extent, the wide variations in income caused by fluctuating zinc and lead prices. Output per manshift on stopes is 12 to 15 tons. Usually nine pairs of stopes provide the tonnage, augmented by development ore, to supply the mill.

Haulage: Haulage drifts and crosscuts are 7x7 ft. Cars are rocker dump, of 25-cu ft capacity. Both Little Trammer and 4-ton locomotives are used, and trains consist of six cars of ore or ten cars of waste. Two locomotives per level service the loading and dumping of stope ore and waste fill. One additional locomotive is required to handle waste and ore from level development. The mine operates two shifts; the afternoon shift goes down while the day shift comes up. All blasting, as nearly as possible, is done at the close of the afternoon shift, leaving 8 hr for full removal of blasting fumes.

Discussion sent (2 copies) to AIME by Sept. 30, 1956, will appear in MINING ENGINEERING and in AIME *Transactions*, Vol. 205, 1956.

Reagent Control in Flotation

Tests at the Sullivan concentrator of The Consolidated Mining & Smelting Co. of Canada Ltd. showed high correlation of metallurgical results to pH and xanthate concentration. A method of analyzing for xanthate in flotation solutions is presented, and the possibility of automatically controlling reagent additions is discussed.

by C. H. G. Bushell and M. Malnarich

REAGENT control in flotation is more an art than a science. Operators vary the amount of reagents used according to the metallurgy obtained. The amount of collector may be increased, for example, if tailings losses are high and decreased if concentrate grades are low. Often control decisions are based on only one assay, such as the tailings analysis by microscopic or *quick assaying* techniques.

This method of control has given reasonably satisfactory results but is clearly far from perfect. No adjustment in reagent addition can be made until information is received that metallurgy has deteriorated, that is, until the optimum time for adjustment has already passed. The first adjustment probably will be an overcorrection or an undercorrection, and time will be required to establish the best new addition rates. The seriousness of the problem depends on variability of the ore. In ore of continually shifting composition and complexity, mill results will shift back and forth from excessively high concentrate grades and high tailings losses to excessively low concentrate grades and low tailings losses. Economic incentive for improved control also increases with increasing size of mill. Even a small improvement in percentage recovery in a large mill pays for expensive control equipment.

The Sullivan concentrator of The Consolidated Mining & Smelting Co. of Canada Ltd. at Kimberley, B. C., treats 11,000 tpd of complex lead-zinc ore, a scale of operations which justifies an extensive research program on automatic control. Work has followed two separate lines: 1) development of faster analytical methods for pulps, and 2) study of metallurgical control by measurement and control of reagent concentrations in flotation solutions.

Although work is incomplete in both phases, a progress report can be given on the development of reagent control.

The primary requirement in flotation is selective attachment of particles of one mineral to air bubbles while the other minerals remain detached. Hydrophobicity of mineral surfaces is established by the collector, which coats the minerals by adsorption. The requirement of selective affinity for air bubbles is therefore equivalent to requiring selective adsorption of a collector. Adsorption theory, supported by abundant experimental evidence, shows that the amount of a reagent adsorbed at a surface must be a function of its residual concentration in solution. As the concentration in solution rises, the amount adsorbed increases, usually following some type of logarithmic curve.

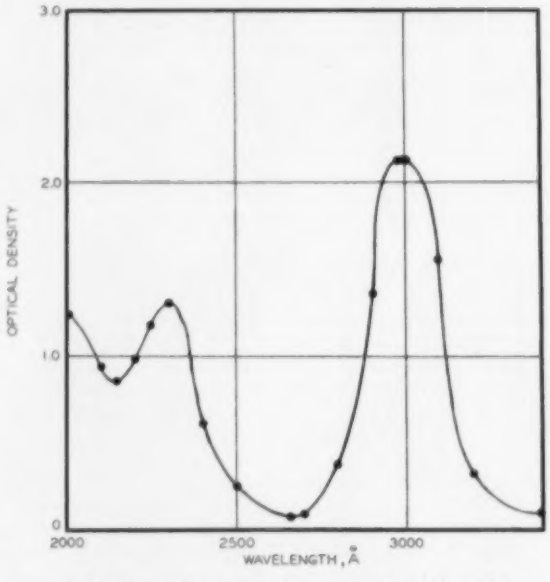


Fig. 1—Spectrum of a solution of potassium isopropyl xanthate.

Selective adsorption must depend on a natural difference in the minerals, but ordinarily the differences in affinity are not great enough to obtain complete coverage of one mineral while the others are completely uncoated. An optimum concentration will exist, below which coverage will not be great enough to float enough of the valuable mineral, and above which too much unwanted mineral will float.

All other reagents, except some frothers, affect the amount of collector adsorbing at constant residual collector concentration in solution, either by adsorbing competitively with the collector, or by a

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TP 4235B. Manuscript, Dec. 5, 1955. New York Meeting, February 1956.

solution effect depending on concentration. Frothing depends upon frother concentration.

Thus the effects of all flotation reagents are direct functions of their residual concentrations in solution. The residual concentrations are related to rates of addition of reagents, but are not perfectly correlated with them because of varying composition of ore, and varying extents of oxidation. (Oxidized compounds are acidic and so affect pH, and may precipitate heavy metal collector salts.) For precise control, residual concentrations must be measured continuously and used to control reagent additions.

The first requirement for reagent control is development of analytical methods that can be used in recording-controllers. The Cominco investigation was carried out specifically for the Sullivan Concentrator, where the following principal reagents are used: for lead flotation, sodium isopropyl xanthate, Dowfroth, lime, and sodium cyanide; for zinc flotation, copper sulfate, sodium isopropyl xanthate, lime, and Dowfroth.

Methods of Chemical Analysis

Xanthate. 1) *Basic Method*: The structural formula for xanthate indicated that its solutions would probably absorb light, and spectrophotometric analysis was therefore suggested. A sample of commercial potassium isopropyl xanthate was purified by dissolving in ethyl alcohol, filtering off the insoluble residue, and precipitating with diethyl ether. The precipitate was then retreated by the same procedure, and the purified xanthate so obtained was washed with ether. Titration with iodine showed the salt to be close to 100 pct pure.

The spectrum of a solution of the purified xanthate was obtained in a Beckman DU spectrophotometer with results shown in Fig. 1. The peak at 3010 A wavelength provided a suitable basis for quantitative analysis. Fig. 2 shows a plot of the height of the peak at 3010 A vs concentration of isopropyl xanthate in solution. Very close correlation results. Both sodium and potassium isopropyl xanthate were used in calibration. Results for equimolar concentrations are identical, as might be expected, since the xanthate part of the molecule is responsible for the absorption of light.

2) *Effects of Impurities*: Spectrophotometric determination of xanthate is practical only if no other substance in the mill solutions affects light absorption at 3010 A. Lime and Dowfroth in solution were found to have no effect on the analysis, but certain other frothers interfere seriously, and the method cannot be used in solutions containing them. Cuprocyanide complex ions interfere slightly, but not to a significant extent. In the absence of copper, sodium cyanide does not affect the analysis. Uncomplexed copper is insoluble at mill pH's and, since solutions must be filtered before analysis, cannot interfere.

Possible reaction products of xanthate, i.e., copper isopropyl xanthate, lead isopropyl xanthate, ferrous isopropyl xanthate, and diisopropyl xanthogen, were synthesized and found to have no significant effect on the determination of xanthate.

Other Reagents: The principal purpose of lime additions is pH control. pH can be measured conveniently with standard commercial instruments. Calcium concentration was also measured to see if lime exerts an effect independent of that on pH.

Analyses for calcium, cyanide, copper, and sulfate were carried out by standard methods of chemical analysis. Some cyanide is converted to thiocyanate in flotation, and it is necessary to distinguish cyanide from thiocyanate. Apart from pH and xanthate, the chemical analyses are not readily adaptable to continuous control. It was decided to proceed with present methods to determine experimentally which reagents are important enough to require continuous control.

No analytical method has yet been developed for Dowfroth. Laboratory tests have shown that Dowfroth does not affect adsorption of xanthate on Sullivan galena.

Mill Tests

In preliminary tests at the Sullivan concentrator, xanthate concentrations were found to be in a range suitable for spectrophotometric determination, being between the extremes of 1 and 100 ppm. Variations in analysis from time to time were appreciable, confirming the desirability of more precise control.

The next step was a series of mill tests to determine correlations between metallurgical results and reagent concentrations. For this purpose one lead rougher (a 16-cell Mineral Separation machine) and one zinc rougher (a 16-cell Mineral Separation machine) were coupled together as a test unit. Feed to the test lead rougher consisted of normal flotation feed (ground fresh feed plus mill returns). Feed to the test zinc rougher consisted exclusively of tailings from the test lead rougher. Concentrates and middlings from the test machines joined the cor-

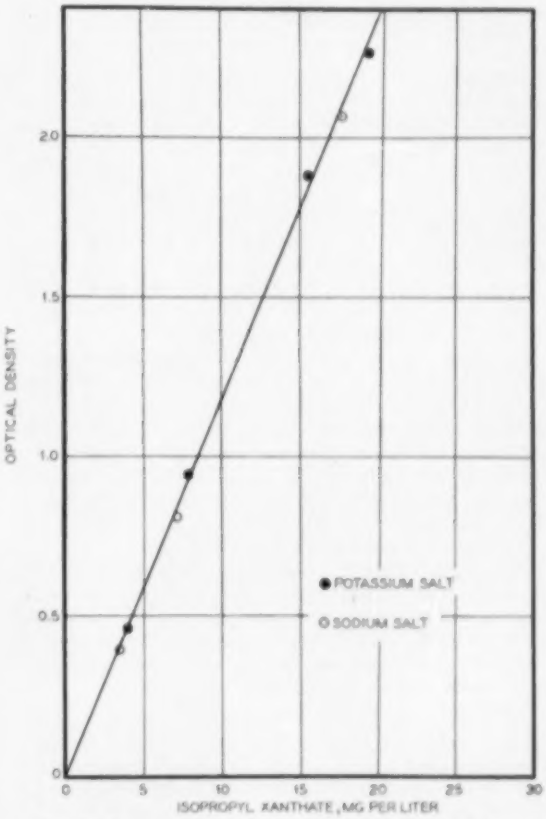

Fig. 2—Concentration of isopropyl xanthate (expressed as xanthate ions) vs optical density at 3010 A.

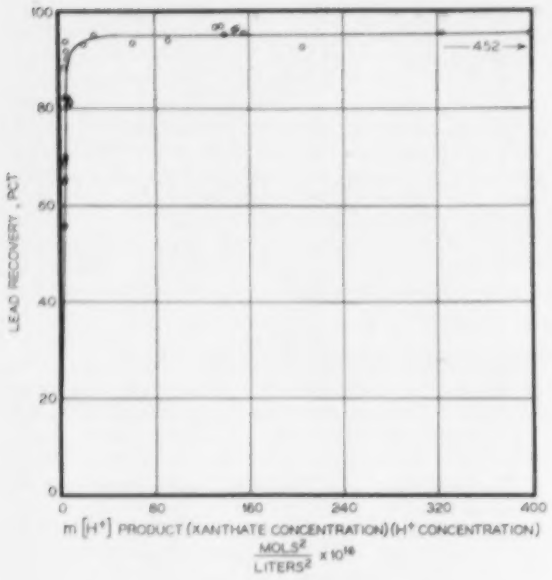

Fig. 3—Effect of xanthate concentration and hydrogen ion concentration on lead recovery in the lead rougher.

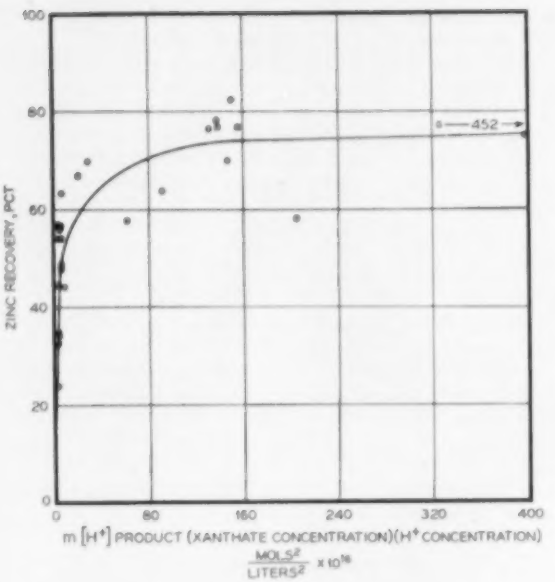

Fig. 4—Effect of xanthate concentration and hydrogen ion concentration on zinc recovery in the lead rougher.

responding products from the other nine lead roughers and nine zinc roughers on normal mill service. With returns from outside the test units joining the flow, optimum reagent concentrations for normal roughing could not be determined. The sole purpose of the test was to determine correlations between metallurgy and residual reagent concentrations.

To explore the situation as quickly as possible, a statistically planned experiment was laid out with reagent concentrations varied over wider limits than in normal practice. Ample data for statistical analysis were obtained, although the experimental design could not be followed because of difficulties in experimental control. No recording instruments were yet available, and feed analyses and tonnages varied sharply, making reagent concentrations difficult to control. To minimize the effects of changing conditions, each test was based on grab samples taken as nearly simultaneously as possible.

Rates of addition of xanthate, lime, and cyanide to the lead rougher and of xanthate, lime, and copper sulfate to the zinc rougher were all varied. Filtered solutions were analyzed for xanthate, pH, total cyanide (including thiocyanate), thiocyanate, calcium, copper sulfate, and reducing power, the last analysis being an empirical test based on iodine titration. Metallurgical balances were calculated from assays of feed and products.

Results were analyzed statistically by both graphical and algebraic methods. The outstanding finding was very high correlation between metallurgy and both pH and xanthate concentration. The amount of copper sulfate added to the zinc roughers has significant effect on zinc metallurgy.

The high correlation between metallurgy and pH and xanthate concentration is easily shown by plotting metallurgical results vs xanthate concentration (m) multiplied by hydrogen ion concentration [H^+].

Figs. 3, 4, and 5 show percentages of lead, zinc, and iron floating in the lead rougher vs the m [H^+] product in lead rougher tailing water. An excellent correlation is obtained for lead and apparently poorer correlations for zinc and iron.

With the methods of calculation used, percentage recoveries with intensive floats were most accurate for lead, next most accurate for zinc, and least accurate for iron. This accounts, in part at least, for the different degrees of scatter. Variation in ore complexity, that is, in number of combined particles, is probably also a factor. At low pH's small errors in pH measurement affect the absolute magnitude of the m [H^+] product appreciably. Tonnage variations contribute slightly to scatter in the zinc curve, as discussed later. Of course, m [H^+] may not be an entirely satisfactory mathematical quantity with which to relate the effects of pH and xanthate on marmatite and pyrrhotite, and the need for an adequate mathematical model of flotation is thereby demonstrated. It must be emphasized that plots of metal recovery against either xanthate or pH separately do not show much correlation. Xanthate concentration and pH must be considered together.

During lead roughing there were extreme ranges in other factors, Table I.

Table I. Factors in Lead Roughing

Tonnage	1000 to 2000 tpd
Pulp density	34 to 54 pct
Pb in feed, pct, including returns	6.7 to 13.2 pct
Zn in feed, pct, including returns	9.0 to 17.4 pct
Fe in feed, pct, including returns	21.4 to 34.3 pct
Total cyanide, including thiocyanate, as equivalent NaCN	10 to 52 ppm
Thiocyanate, as equivalent NaCN	9 to 30 ppm
Calcium (expressed as CaO)	200 to 1000 ppm
SO_4	500 to 1100 ppm
Reducing power	220 to 550 ml N/100 L_2 per liter

Graphical correlation indicated that none of the foregoing factors affected percentage recovery except tonnage, which appeared to have a slight effect on the amount of zinc floating. Of course, some of the variables might affect percentage recovery if varied outside the range of the existing tests. Furthermore, small effects within the range of the exist-

ing tests could escape detection by graphical correlation analysis. There is no doubt, however, that pH and xanthate concentration have dominant importance within the range of test conditions.

Results for flotation in the zinc rougher were similar, but correlations were lower, largely because the results were affected by conditions in the lead rougher.

Discussion

The dependence of metallurgy on the m [H^+] product deserves attention. Wark[1] quotes Barsky as drawing attention to the fact that critical pH curves as determined by captive bubble tests follow the relationship

$$[X^-]\,[H^+] = \text{constant}. \qquad [1]$$

Here [X^-] = concentration of xanthate ions and [H^+] = concentration of hydrogen ions. At mill pH's xanthic acid is almost completely dissociated, so that m = [X^-].

M. A. Cook et al. have written a series of papers advocating the free acid collector theory.[2-4] According to this theory, the sole effect of pH is to control the degree of dissociation of xanthic acid. The equilibrium expression for xanthic acid is:

$$\frac{[H^+]\,[X^-]}{[HX]} = K_a. \qquad [2]$$

Here K_a is the dissociation constant.

The spectrophotometer measures the total xanthate concentration

$$[HX] + [X^-] = m. \qquad [3]$$

Hence

$$\frac{[H^+]\,\{m - [HX]\}}{[HX]} = K_a$$

which may be rearranged to

$$[HX] = \frac{m\,[H^+]}{K_a + [H^+]}. \qquad [4]$$

For isopropyl xanthate at mill pH's, $K_a >> [H^+]$. Hence, as an adequate approximation,

$$[HX] = \frac{1}{K_a} \cdot m\,[H^+]. \qquad [5]$$

According to the Cook theory, therefore, the action of pH and xanthate concentration on flotation can be described by the quantity $m[H^+]$.

The importance of the $m[H^+]$ product can be regarded also as a special case of an equation resulting from competitive adsorption between xanthate ions and hydroxyl ions. The simplest equation for competitive adsorption has the form

$$\theta = \frac{m}{m + a[OH^-] + b} \qquad [6]$$

where θ = fraction of available sites on the mineral surface occupied by xanthate, m = concentration of xanthate, a = constant, and b = constant.

Mineral recovery must be a function of θ, but

$$[H^+]\,[OH^-] = K_w \qquad [7]$$

where K_w = dissociation constant of water. Hence

$$\theta = \frac{m}{m + \dfrac{aK_w}{[H^+]} + b} = \frac{m[H^+]}{m[H^+] + aK_w + b[H^+]}. \qquad [8]$$

Flotation recovery is a function of $m[H^+]$ when b is small enough to make the term $b[H^+]$ negligible.

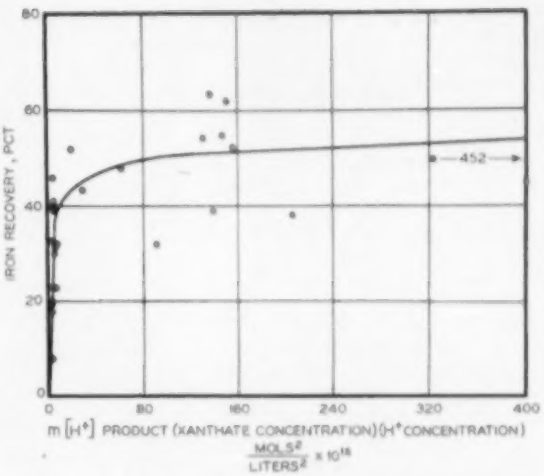

Fig. 5—Effect of xanthate concentration and hydrogen ion concentration on iron recovery in the lead rougher.

Of course, more complex adsorption equations may actually be required. The important point is that high correlation between metallurgy and $m[H^+]$ does not in itself constitute proof as to whether adsorption of undissociated xanthic acid or competitive adsorption between xanthate ions and hydroxyl ions is the correct mechanism.

As a result of the high correlation between mill metallurgy and xanthate concentration and pH, two test recording instruments—one pH meter and one xanthate meter—are on order to further the development of automatic control in the Sullivan concentrator.

In order to take full advantage of reagent control, the action of the various reagents must be fully understood. At present, adsorption tests are being run to determine the effects of various reagents on the adsorption of xanthate on galena, marmatite, and pyrrhotite. This work is supplemented by laboratory flotation tests in which all solutions are analyzed for residual reagent concentrations. Progress has been made in developing a mathematical model of flotation to facilitate interpretation of data.

Acknowledgments

This work is a cooperative venture between the Research and Development Div. and the concentration department of the Mines Div. of The Consolidated Mining and Smelting Co. of Canada Ltd. In the Research and Development Div., J. R. Mills, K. Raht, J. N. Robinson, H. L. Williams, W. L. M. Phillips, and R. R. McLaughlin contributed in various ways. Particular thanks are due H. Greenspan, who did the early laboratory work on development of analytical methods. In the concentration department, the excellent cooperation of A. G. Robertson and P. M. Elliott made the work possible.

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A Kinetic Study of the Leaching of Molybdenite

A study of the rate of dissolution of molybdenite (MoS_2) in alkaline solution was carried out under carefully controlled conditions. Effects of temperature, oxygen over-pressure, and KOH concentration were evaluated. Studies were made in the temperature range 100°C to 175°C and in the pressure range 0 to 700 psia of oxygen. Under these conditions molybdenite was found to leach according to a linear mechanism. Both oxygen over-pressure and KOH concentration were found to control the rate of leaching. The mechanism has been explained in terms of adsorption of oxygen at the molybdenite surface followed by configurational rearrangement of the adsorbed molecules. The hydroxyl ion dependency is believed to be diffusion-controlled. Laboratory batch studies have shown that molybdenite can be readily dissolved in alkaline solutions under moderate conditions of temperature and pressure. Commercial application of this process to the production of ferro-molybdenum and molybdenum chemicals is promising in view of the ease of dissolution of molybdenite and the relatively noncorrosive conditions involved in the process.

by William H. Dresher, Milton E. Wadsworth, and W. Martin Fassell, Jr.

HIGH temperature-high pressure techniques have long been used to great advantage in the organic chemical industry, the petroleum industry, and the paper industry. Only recently, however, have these methods been used to extract metals from their ores on a commercial scale. The Chemical Construction Corp., together with interested producers of nickel and cobalt, has done much to develop methods of producing nickel and cobalt powders by the use of high temperature-high pressure techniques.[1] The Howe Sound Co. is currently operating a plant near Salt Lake City in which the new Chemical Construction Corp. technique is applied to cobalt-arsenic sulfide concentrates. In this method the concentrates are leached in a dilute solution of sulfuric acid under air pressure at a high temperature. Near Edmonton, in Alberta, Canada, the Sherritt Gordon Co. is also using the new method in conjunction with its ammoniacal leach process.[2] National Lead Co., at Fredricktown, Mo., is operating a process similar to that of Howe Sound.

In spite of the rapid growth of high temperature-high pressure hydrometallurgical processes, there has been very little work done on fundamental principles involved in this type of process. Consequently the literature lacks any significant data in this field. Oxidation of pyrite,[3] sphalerite,[4] galena,[5] and recently molybdenite[6] in alkaline solutions has been reported to some extent.

This investigation was initiated, therefore, to broaden the application of high temperature-high pressure hydrometallurgy and also to contribute some insight into the basic mechanisms of oxidation in an aqueous medium under these conditions. In addition to the immediate consequences of these processes much valuable information remains to be learned about fundamental reactions when they are carried out at temperatures and pressures slightly higher than those normally encountered.

Leaching of molybdenite by alkaline solutions under oxygen pressure has recently been reported in the Russian literature by E. S. Usataya,[6] who found that leaching rate increased with an increase in the pH of the solution—the strongest effects being observed at a pH of 10—and also with an increase in temperature. Usataya also has reported the formation of a protective film on the surface of the molybdenite when the leaching solution was weakly alkaline, neutral, or acidic. He believes that this oxidation process explains the migration of Mo^{++} in the oxidation zones of molybdenite ore deposits and the subsequent impoverishment of the ores.

Equipment: The high temperature-high pressure reaction unit used in this work was especially designed for kinetic studies of this nature, emphasis being placed on accuracy of measurement and precision of control. As details of the unit's construction and operation have been discussed elsewhere,[7] only basic features and principles of operation will be discussed at this point.

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TP 4243BD. Manuscript, Sept. 14, 1955. New York Meeting, February 1956.

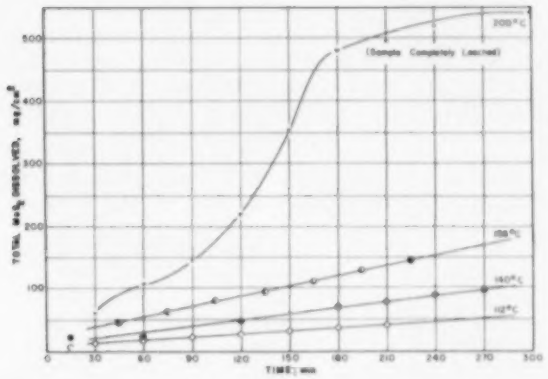

Fig. 1—Typical rate curves.

1) Temperature of the solution within the autoclave is constantly measured, recorded, and controlled.

2) Gas over-pressure of the system is constantly regulated and indicated.

3) Provisions are made for lowering a sample into the reaction zone of the autoclave once thermal equilibrium has been attained.

4) A means is provided for removal of liquid samples from the reaction zone of the autoclave during the course of the reaction without disrupting operation of the unit.

Solubility of oxygen in various concentrations of solutions used in this experiment was measured in equipment adapted from that described by Frolich, et al.[8] and by Ipatieff and Monroe.[9] The oxygen-saturated solution from the autoclave was admitted to the gas-measuring equipment by means of a water-cooled capillary tube. A vacuum-tight connection was made between the capillary tube and the flash chamber of the gas-measuring equipment. The liquid was measured by a standard 100-ml capacity burette encased in a water jacket, and the volume of gas was measured in a Fisher precision gas burette.

Bench-scale tests were made in an Autoclave Engineers 1-gal, electrically heated autoclave. A Leeds and Northrup type C Micromax controller was used to control temperature of the solution within the autoclave.

Experimental Procedure: Molybdenum sulfide used in the kinetic study was Molysulfide lubricant, Grade 2, a commercial product manufactured by Climax Molybdenum Co. The MoS_2 content of this material was better than 99 pct. A less pure grade of molybdenite, obtained from the Kennecott Copper Co., was used for the bench-scale tests. This material contained approximately 89 pct MoS_2, the remainder being a siliceous gangue material.

Solid molybdenite samples, 0.650 in. diam and approximately 0.085 in. thick, were prepared by pressing the powdered molybdenite (97 pct, −100 mesh) in a Carboloy die at a pressure of approximately 85 tons per sq in. These samples varied from 91 to 92 pct of theoretical density of massive molybdenite. The geometric surface area of the molybdenite disk was calculated from micrometer measurements of the sample. After preparation, the molybdenite disks were pressed by hand into an Inconel sample holder and mounted in the autoclave. Flat, plate-like samples were used in these studies in order to maintain the geometric surface area essentially constant during the course of the run. Rate curves for the dissolution of MoS_2 powders were found to deviate from linearity because of the large variation in surface area as the reaction progressed.

Molybdenum content of the leach solution, measured at periodic intervals, was determined spectrophotometrically from a phenylhydrazine hydrochloride-molybdenum complex by a Beckman Model DU quartz spectrophotometer. The method of analysis followed that developed by Ayres and Tuffly.[10]

Measurements of oxygen solubility in the leach solutions were made with the equipment previously described. Since a relatively large volume of solution was required for each solubility measurement, it was not possible to measure solubility concurrently with molybdenum concentration. When a solubility measurement was made, the solution within the autoclave was brought to the desired temperature and the oxygen over-pressure was increased in increments of 100 to 200 psi. A measurement of the amount of oxygen dissolved in 50 ml of the leach solution was determined for each increment. Each measurement consisted of a reading of the volumes of both liquid and gas, of the temperature of the gas in the burette, and of the atmospheric pressure. Results of these measurements were expressed in mols of oxygen per liter of solution. Solubility measurements were made for all temperatures and KOH concentrations used in the leaching studies.

Batch studies on leaching molybdenite were carried out in the Autoclave Engineers 1-gal autoclave. In this series of studies, potassium hydroxide solution and powdered molybdenite were charged to the

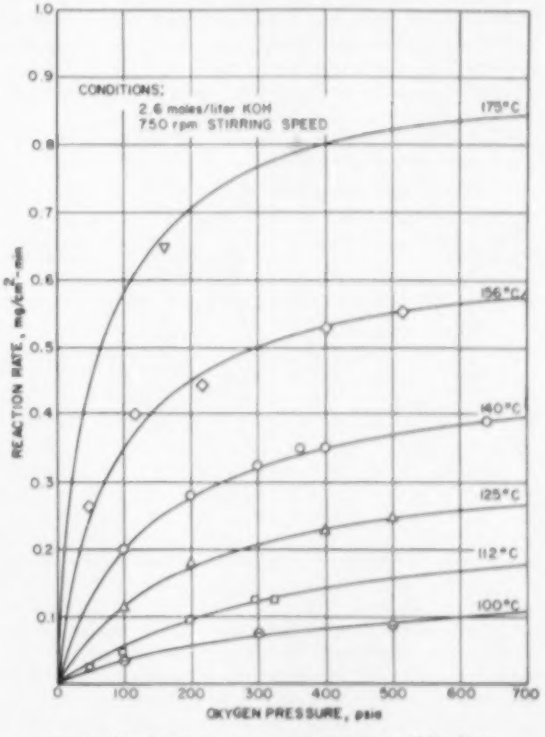

Fig. 2—Effect of oxygen over-pressure on rate of leaching.

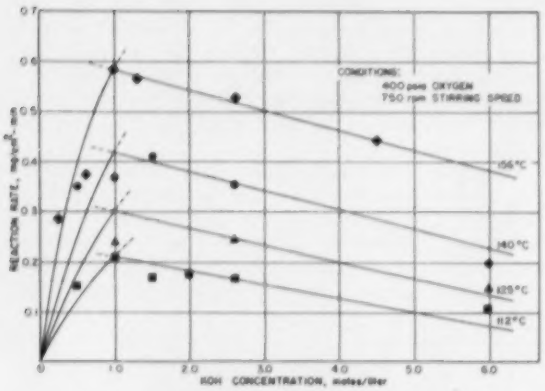

Fig. 3—Effect of KOH concentration on rate of leaching.

autoclave, which was evacuated and allowed to come to operating temperature without agitation. Oxygen was admitted to the autoclave to the desired working pressure and agitation started. Samples of the solution were withdrawn at intervals by means of a capillary tube. Due to the high molybdenum concentration of the solution the progress of the reaction was followed by titrating the KOH remaining in the solution rather than by the colorimetric method of analysis used in the kinetic study.

Experimental Results: The reaction rate data consisted of measurements of the rate of formation of MoO_4^{--} under the various conditions of leaching. Results to be described are based on the leaching of more than 65 samples of molybdenite in potassium hydroxide solution under oxygen partial pressure of 0 to 700 psia and at temperatures from 100°C to 175°C. As shown in Fig. 1, molybdenite was found to leach according to a linear mechanism under nearly all conditions investigated. When relatively high rates of reaction were involved, as in runs at 175°C or higher, deviation from linearity was observed. This deviation was found to be due to disintegration of the pressed samples and a subsequent increase in effective surface area.

Table I. Bench-Scale Tests of Molybdenite Leaching

Oxygen Pressure, Psia	Temperature, °C	Molar Ratio, MoS_2:KOH	Time to 95 Pct Completion, Min
200	156	1:2	62
400	156	1:2	38
600	156	1:2	33
800	156	1:2	20
1000	156	1:2	22
1500	156	1:2	17
400	108	1:2	110
400	125	1:2	85
400	140	1:2	65
400	156	1:2	38
400	175	1:2	23
400	156	1:4	28
400	156	1:2	38
400	156	4:5	46

Four variables were studied in determining kinetics of the leaching reaction: *speed of agitation, KOH concentration, temperature,* and *oxygen over-pressure.* Rate of the reaction was grossly affected by speed of agitation below approximately 500 rpm. Above this speed, however, reaction rate was relatively stable until cavitation occurred at about 1000 rpm and the sample disk was no longer covered by the leach solution. Stirring rate was maintained at 750 rpm for all tests. Fig. 2 shows the experimental rates of leaching for various pressures of oxygen, Fig. 3 the effect of a change in the leachant concentration on the rate of leaching for several different temperatures.

A series of bench-scale tests were carried out to demonstrate commercial feasibility of the leaching process. Optimum conditions were taken from the results of the kinetic studies and briefly studied batchwise. Oxygen over-pressure, temperature, and various molar ratios of MoS_2 to KOH were studied. Results of these tests, expressed as the time required to leach 95 pct of the molybdenite, are presented in Table I.

Tests similar to those presented for KOH solution were carried out in NaOH and Na_2CO_3 solutions. The reaction was found to proceed using each of these reagents. The rate measured in Na_2CO_3 solutions was much slower because of the buffering action of Na_2CO_3 and the resulting lower hydroxyl ion concentration. Use of sodium salts produces Na_2MoO_4, which is less soluble than K_2MoO_4—115.5 g per 100 ml at 100°C for Na_2MoO_4 as compared to 184.6 g per 100 ml at 25°C for K_2MoO_4.

Discussion of Results

The linear reaction rates observed for all experiments, Fig. 1, carried out within the range of the conditions of this study indicate that the surface is not hindered by products of the reaction. Any build-up of adherent surface layers as a result of the reaction would result in nonlinear rate curves. The linearity of the rate curves also indicates that the concentration of surface sites that contribute to the reaction remains constant for any given set of conditions.

The overall measured rate is clearly a function of the oxygen over-pressure, as is shown in Fig. 2. The shape of this curve suggests a surface adsorption process that reaches a maximum rate for complete surface coverage. The oxygen dependency curves are regular and typical of a process involving adsorption of a gas onto a surface. Hydroxyl ion dependency, however, is not so simple. The isotherms in Fig. 3 can be divided into two regions. In the first region, below a concentration of approximately 1.0 mols per liter, the reaction rate is an increasing function of the hydroxyl ion concentration. Above this region the rate decreases with increasing hydroxyl ion concentration. This rather strange shape of the KOH dependency may be attributed to three major effects: 1) the concentration of oxygen dissolved in the solution varies with the KOH concentration for any given over-pressure, 2) the activity coefficient of KOH varies considerably over the range of concentrations of KOH considered, and 3) the combined effect of diffusion and viscosity must be taken into consideration. To eliminate the first of these effects the solubility of oxygen in KOH solutions must be known. Since the literature lacks data obtained under the conditions of this study these measurements were made experimentally. Fig. 4 shows the solubility of oxygen in KOH solutions under the conditions employed in each of the leaching studies. The quantity S/P is the ratio of solubility of oxygen in mols per liter to the absolute pressure of oxygen over the solution. Activity coefficients of the various concentrations of KOH solutions were taken from data of Harned and Cook[13]

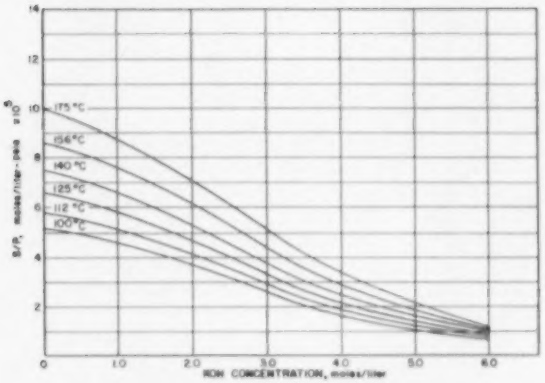

Fig. 4—Solubility of oxygen in KOH solutions.

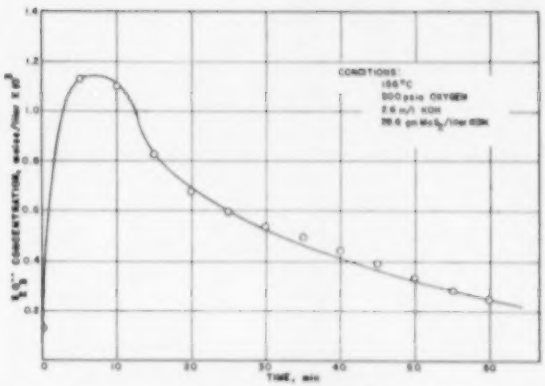

Fig. 5—Formation of S_2O_3 during leaching of MoS_2.

and extrapolated to the desired temperatures. Since the concentrations of oxygen in solution were so low—10^{-4} mols per liter—activity coefficients of the oxygen dissolved in the KOH solutions were assumed to be equal to unity.

The overall reaction may be explained stoichiometrically by Eq. 1:

$$MoS_2 + 9/2O_2 + 6OH^- \rightarrow MoO_4^{--} + 2SO_4^{--} + 3H_2O. \quad [1]$$

After complete oxidation the end products were the molybdate and the sulfate ions. If, however, the solutions were analyzed as soon as they were removed from the autoclave and the molybdenum sulfide was not completely reacted, the sulfur was found to be present in solution as the sulfate ion plus a measurable quantity of the thiosulfate ion —$S_2O_3^{--}$. This was determined by a standard iodine titration. Fig. 5 is a typical plot of $S_2O_3^{--}$ ion concentration vs time. The presence of the thiosulfate ion in reactions of this type has been reported previously by Forward et al.,[12] who found curves of this type for the dissolution of cobalt, nickel, and copper sulfides in ammoniacal solutions. They also noted the presence of dithionate ion and the higher thionates as well as sulfamate. However, thionates cannot exist for any appreciable time in alkaline solutions in the absence of the ammonia ion due to their decomposition by the hydroxyl ion resulting in the formation of thiosulfate ion.[13] The direct formation of the thiosulfate ion may be represented by the intermediate overall reaction

$$MoS_2 + 5/2O_2 + 4OH^- \rightarrow MoO_4^{--} + S_2O_3^{--} + 2H_2O. \quad [2]$$

The complexity of these reactions (Eqs. 1, 2) requires a series of reactions, of which one or more may be rate-controlling. In any event, it appears that the surface reactions involved result in intermediate compounds that must be capable of forming the observed end products and at the same time maintain the proper oxygen and charge balance.

Theoretical

Development of the Rate Equation: According to the Absolute Reaction Rate Theory,[14] the rate of any chemical reaction may be represented by the general equation

$$\text{Rate} = \sum_j \pi_{ij} c_{ij} k'_j = \sum_j \kappa \frac{kT}{h} \pi_{ij} c_{ij} e^{-\Delta F_j^+/RT} \quad [3]$$

where $\sum_j$ represents the summation of all contributing rates, $\pi_{ij} c_{ij}$ represents the product of the reactant concentrations for the j^{th} rate step, k'_j is the specific rate constant, κ is the transmission coefficient, ΔF^+_j is the activation energy of the j^{th} rate step and k and h are the Boltzman and Planck constants, respectively. Application of Eq. 3 requires evaluation of the rate-contributing steps involved as well as evaluation of the reactant concentrations $\pi_{ij} c_{ij}$ for each step.

The following steps are suggested as possibly leading to the overall reaction commensurate with the measured rates and the observed end products:

$$|MoS_2 + O_2 \rightarrow |MoS_2 \ldots O_2 \quad \text{(fast)} \quad [4]$$

$$|MoS_2 \ldots O_2 + O_2 \underset{k_2}{\overset{k_1}{\rightleftharpoons}} |MoS_2 \ldots 2O_2 \quad \text{(slow)} \quad [5]$$

$$|MoS_2 \ldots 2O_2 \xrightarrow{k_3} \left[MoS_2 \ldots 2O_2\right]act \quad \text{(slow)} \quad [6]$$

$$\left[MoS_2 \ldots 2O_2\right]act + OH^- \rightarrow |MoO_3(OH)^- + S_2O_3^{--} \quad \text{(fast)} \quad [7]$$

$$|MoO_3(OH)^- + OH^- \rightarrow MoO_4^{--} + 2H^+ \quad \text{(fast)} \quad [8]$$

$$H^+ + OH^- \rightarrow H_2O \quad \text{(fast)} \quad [9]$$

$$S_2O_3^{--} + 1/2\,O_2 \rightarrow S_3O_6^{--} \quad \text{(fast)} \quad [10]$$

An alternate mechanism would involve adsorption of a single molecule of O_2 as in Eq. 4 and the subsequent formation of S_3^{--} in Eq. 7.

The symbol | is used to denote a surface site or product of reaction at the surface. Electrical neutrality must of course be maintained by the presence of gegen ions of the proper charge. It is not intended in these reactions to indicate how oxygen is adsorbed on the active site. The character of the curves of Fig. 2 indicates molecular adsorption. This does not necessarily imply physical adsorption, since the molecular oxygen may possibly split on the sulfur

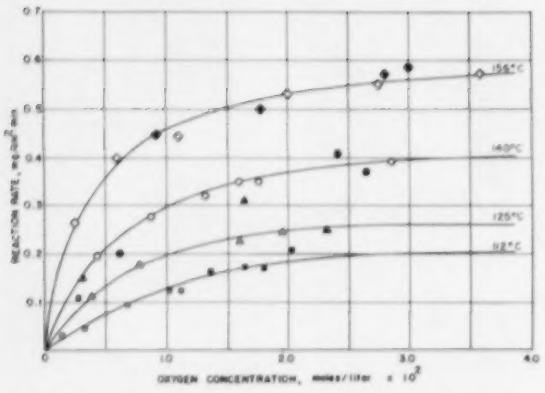

Fig. 6—Demonstration of the independence of KOH concentration.

or in some way split between the surface molybdenum and the sulfur. Kinetically, such splitting will appear as molecular adsorption. Eq. 6 represents a step in which the adsorbed oxygen undergoes a configurational change creating an active site prone to the action of the hydroxyl ion. The molybdenum oxy-hydroxide ion, $MoO_2(OH)^+$, proposed in Eq. 7 is not unlikely, since molybdenum oxy-sulfate, $MoO_2(SO_4)$, and MoO^{++}, are known.[15] The intermediate thiosulfite ion, $S_2O_3^{--}$, or perhaps 2 SO^-, provides a proper step in terms of oxygen and charge balance to explain the end products observed. These two intermediates would be very unstable and would react immediately at or near the surface according to Eq. 10. The final presence of sulfate ion, SO_4^{--}, results from further oxidation of thiosulfate in basic solution according to the reaction

$$S_2O_3^{--} + 2\,O_2 + 2\,OH^- \rightarrow 2\,SO_4^{--} + H_2O. \quad [11]$$

The addition of this reaction results in Eq. 1.

The present development is based upon a steady-state treatment in which the concentration of surface reactant is considered to be constant for any one set of conditions. These data may be treated in a similar fashion by considering surface equilibrium for adsorption of oxygen. Arguments will be presented, however, which indicate that equilibrium was not attained. If the fraction of total available sites covered with oxygen is considered (i.e., $MoS_2 \ldots 2O_2$ sites) as θ, under the steady-state treatment rate of formation of the $MoS_2 \ldots 2O_2$ sites is just balanced at the rate at which they disappear.[16] Accordingly, from Eqs. 5 and 6

$$\frac{d\theta}{dt} = k_0(1-\theta)\,[O_2]\,k_1 - \theta k_2 k_0 - \theta k_0 k_3 = 0. \quad [12]$$

The quantity $(1-\theta)$ represents the fraction of the available surface present as $MoS_2 \ldots 2O_2$. This implies that reaction (Eq. 4) is so fast that the fraction of the available surface sites completely uncovered by oxygen (MoS_2) is negligible. The constant k_0 contains the terms necessary to convert from surface coverage to the true surface concentration, surface roughness, and the fraction of the number of surface sites that are active. The constants k_1 and k_2 represent the specific reaction rate constants for the forward and the backward reaction, respectively, of Eq 5. The constant k_3 represents the specific reaction rate constant for the reaction of Eq. 6. From Eq. 12 the value of θ may be written in typical Langmuir form:

$$\theta = \frac{\frac{k_1}{k_2 + k_3}\left[O_2\right]}{1 + \frac{k_1}{k_2 + k_3}\left[O_2\right]}. \quad [13]$$

Rate of the reaction was experimentally determined by the rate of the appearance of MoO_4^{--} in solution. Since the reaction represented by Eq. 7 is very fast, the rate of formation of MoO_4^{--} is determined by the rate of formation of the intermediate $MoO_2(OH)^+$ according to Eq. 6. Therefore,

$$\text{Rate} = \frac{d(MoO_4^{--})}{dt} = k_0\,\theta\,k_3. \quad [14]$$

Combining Eqs. 13 and 14 results in Eq. 15:

$$\text{Rate} = \frac{k_0 k_3 \frac{k_1}{k_2 + k_3}\left[O_2\right]}{1 + \frac{k_1}{k_2 + k_3}\left[O_2\right]} \quad [15]$$

which, at constant temperature, can be written as Eq. 16:

$$\text{Rate} = \frac{k_a k_b\,[O_2]}{1 + k_b\,[O_2]} \quad [16]$$

where $k_a = k_0 k_3$ and $k_b = k_1/(k_2 + k_3)$. This represents the conditions of the tests presented in Fig. 2. Each curve was determined at constant temperature and at a constant KOH concentration of 2.6 mols per liter. Solid lines drawn through the experimen-

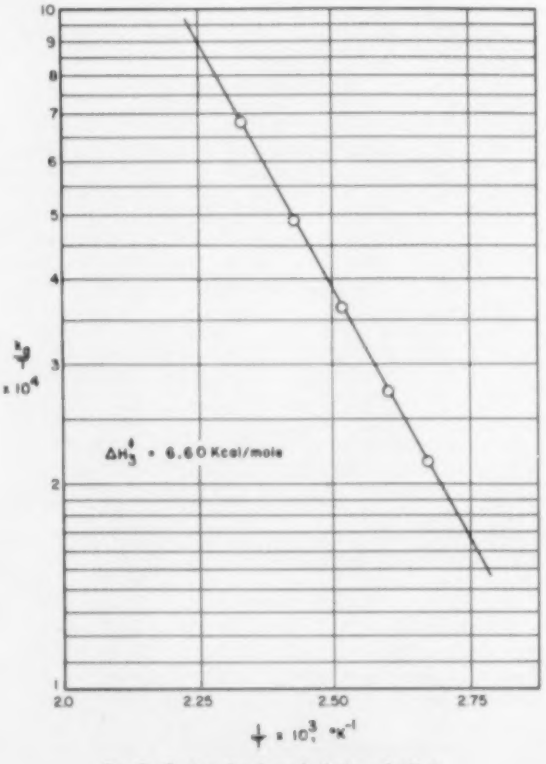

Fig. 7—Eyring plot for calculation of $\Delta H_3^{\ddagger}$.

tal points of Fig. 2 are the theoretical calculated curves according to Eq. 16. Plateau values of the curves represent complete surface coverage or $\theta \rightarrow 1$. These plateau values therefore represent the k_a terms at each temperature. From this and one other point on each curve the value of k_b may be determined for each isotherm according to Eq. 16. The term $k_1/(k_a + k_2)$ approaches the equilibrium constant K as k_2 becomes small compared to k_b. Thus, $k_b \rightarrow K$ and then represents the equilibrium adsorption of oxygen in Eq. 5. From the curves of Fig. 2 the values of k_b were found to be 234, 300, 378, and 480 at the temperatures of 112°, 125°, 140°, and 156°C respectively. Since all surface adsorption processes must be exothermic, i.e., K must decrease with an increase in temperature, this increase in the value of k_b with temperature indicates that equilibrium was not attained. For this reason, the second alternative $k_2 >> k_a$ must be chosen and the steady-state treatment adopted.

The effect of varying hydroxyl ion concentration is irregular from what might be expected. The curves of Fig. 3 were determined for various KOH concentrations at a constant over-pressure of oxygen of 400 psia and a constant stirring speed of 750 rpm. The concentration of oxygen dissolved in the KOH solution was not a constant due to the variation of the solubility of oxygen in the various concentrations of KOH, as indicated in Fig. 4. The effect of varying the hydroxyl ion concentration can be explained by considering the two sections of each of the isotherms separately. In the first section the solubility of oxygen in the dilute KOH solution is sufficient to supply an adequate amount of oxygen to the surface of the molybdenite in order to support the reaction. In this section, however, the hydroxyl ion does not diffuse to the surface fast enough to continue the reaction. Thus the rate of the reaction increases as the hydroxyl ion is increased. At a concentration of approximately 1.0 mols per liter the rate of supply of hydroxyl ion to the surface becomes equal to the rate of formation of active oxidized surface sites. Beyond this point, as the hydroxyl ion concentration continues to increase, oxygen solubility is limited to such an extent as to hinder the reaction by not supplying a sufficient amount of oxygen to maintain the oxidation reaction at an equal or faster rate than the reaction with the hydroxyl ion. In this section of the curves the rate thus becomes zero order with respect to KOH, since the overall rate now depends on the rate of formation of active oxidized surface sites which are removed by reaction with hydroxyl ion as soon as they are formed. The only hydroxyl dependency is manifest by the solubility of the oxygen in the KOH solution. This last point is clearly demonstrated by the fact that it is possible to intermix the experimental data of Fig. 2, determined at a constant KOH concentration of 2.6 mols per liter and at various oxygen over-pressures, with the hydroxyl ion independent data (above 1.0 mols per liter KOH) of Fig. 3 determined at various KOH concentrations and at a constant oxygen over-pressure of 400 psia. These data are shown in Fig. 6 where the rate of the reaction is plotted vs the concentration of oxygen dissolved in the KOH solutions. The data indicated by the open symbols are those obtained from Fig. 2; data indicated by the closed symbols are obtained from Fig. 3.

Thermodynamic Considerations: Using the steady-state treatment and the Absolute Reaction Rate

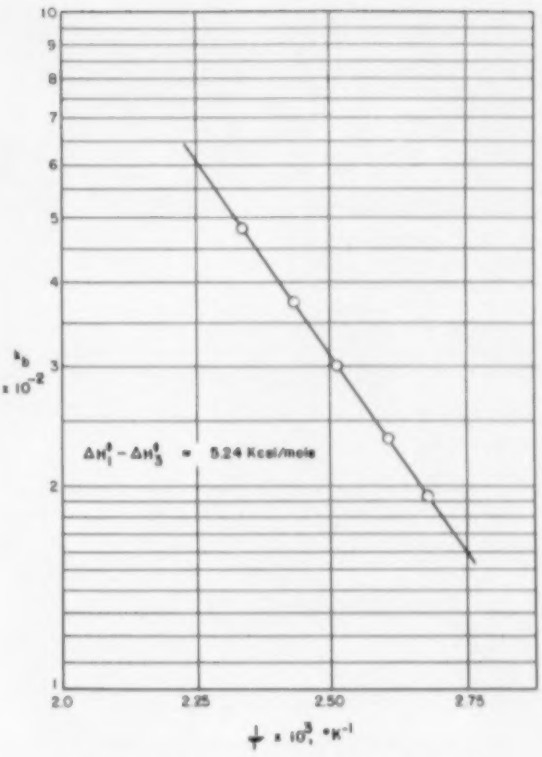

Fig. 8—Eyring plot for calculation of $\Delta H_1^{\ddagger} - \Delta H_2^{\ddagger}$.

Theory,

$$k_a = k_a k_c = k_a \exp\left[-\Delta F_a^{\ddagger}/RT\right] = k_a \exp\left[-\Delta H_a^{\ddagger}/RT\right] \exp\left[\Delta S_a^{\ddagger}/R\right]. \quad [17]$$

The slope of the Eyring plot of ln k_a/T vs $1/T$ should give the value of $\Delta H_a^{\ddagger}$. Fig. 7 represents such a plot, from which it was found that $\Delta H_a^{\ddagger}$ is equal to 6.60 kcal per mol. This represents the heat of activation for reaction (Eq. 6) which involves rearrangement of the adsorbed oxygen on the surface. Likewise, it can be shown that

$$k_b = \frac{k_1}{k_2} = \exp\left[-(\Delta F_1^{\ddagger} - \Delta F_2^{\ddagger})/RT\right] = \exp\left[-(\Delta H_1^{\ddagger} - \Delta H_2^{\ddagger})/RT\right] \exp\left[(\Delta S_1^{\ddagger} - \Delta S_2^{\ddagger})/R\right]. \quad [18]$$

The slope of the Eyring plot ln k_b vs $1/T$ gives a value of $\Delta H_1^{\ddagger} - \Delta H_2^{\ddagger}$. Fig. 8 shows that $\Delta H_1^{\ddagger} - \Delta H_2^{\ddagger}$ is equal to 5.24 kcal per mol. By difference, $\Delta H_1^{\ddagger}$ is found to be equal to 11.84 kcal per mol. This represents the heat of activation for adsorption of oxygen onto the surface of the molybdenite according to Eq. 5.

It is interesting to make an approximation of the number of surface sites actually involved in the leaching process. If a rate is selected where $\theta = 1$, Eq. 5 can be used to calculate k_a, which contains the factors: k_{ss}, surface roughness; k_{ACT}, the fraction of the total number of surface sites activated; and k', the geometric number of molecules per unit area of surface. Calculation of the number of MoS_2 sites on the principle cleavage plane of molybdenite—

the 0001 plane—gives the value of approximately 4.95×10^{15} mols per cm². The difference in entropies $\Delta S_1^{\ddagger} - \Delta S_s^{\ddagger}$ can be calculated from the relationship

$$\ln \frac{k_1}{k_s} = -\frac{(\Delta H_1^{\ddagger} - \Delta H_s^{\ddagger})}{RT} + \frac{(\Delta S_1^{\ddagger} - \Delta S_s^{\ddagger})}{R} . \quad [19]$$

This gives a value of 24.45 eu per mol. $\Delta S_1^{\ddagger}$, the entropy change associated with the adsorption of oxygen onto the surface of the molybdenite, can be estimated by the use of the B.E.T. assumption that the energy of adsorption is approximately equal to the energy of liquifaction of adsorbed gas. The free energy of activation for each of the processes can be calculated by the thermodynamic relationship

$$\Delta F = \Delta H - T\Delta S . \quad [20]$$

Table II shows the percentage of active sites for several estimations of $\Delta S_1^{\ddagger}$ calculated at the conditions: $T = 429°K$, $P = 400$ psia ($[O_2] = 2.0 \times 10^{-4}$ mols per liter), KOH = 2.6 mols per liter ($[a_{OH^-}] = 2.34$), and Rate = 0.530 mg per cm². min (3.33×10^{15} mol per cm². sec). Values of $\Delta S_1^{\ddagger} = -6.00$ eu per mol and $k_{ss} = 10$ may be selected as being nominal estimates of the entropy of adsorption and of the surface roughness factor. In this case, the number of active sites is approximately 0.26 pct of the number of total surface sites. This suggests that the face of a molybdenite platelet is relatively inactive and that the reaction probably occurs at the edges of these platelets, removing MoS_2 in a series of steps.

Table II. Estimation of the Number of Active Surface Sites

$\Delta S_1^{\ddagger}$ Eu per Mol	$\Delta S_s^{\ddagger}$ Eu per Mol	k_{ss} Mols Per Cm²	$k_{ss}k_{ACT}$	Active Sites, Pct $k_{SS}=1$	Active Sites, Pct $k_{SS}=10$
0	−24.45	1.7×10^{13}	3.4×10^{-4}	0.034	0.0034
−6.00	−30.45	1.3×10^{14}	2.6×10^{-3}	2.6	0.26
−12.00	−36.45	7.9×10^{14}	1.6×10^{-1}	16.0	1.60

Commercial Application

The product of the leaching process is a pregnant liquor carrying the desired molybdenum plus any other alkali-soluble metals that may have been present in the ore. Such a process, if applied to molybdenite ores containing appreciable quantities of rhenium, would result in a liquor containing potassium perrhenate as well as potassium molybdate. This liquor could then be treated by ion exchange or by solvent extraction to separate the rhenium from the molybdenum.

The sands flotation treatment in the molybdenum circuit, such as is currently used to separate molybdenum sulfide from the underflow of copper flotation circuits, could be eliminated by application of this process for dissolution of molybdenum. The depressed molybdenite plus the gangue material removed following the bulk sulfide step should be directly applicable to such a leaching process.

A novel application of this high temperature leach process has been devised in which use is made of waste pickle liquors to produce ferro-molybdenum for steel alloying. A feature of this process is the fact that the leachant can be recovered by treating the spent leach solution with hydrated lime. The calcium sulfate formed is filtered off, leaving the desired potassium hydroxide. The ferric molybdate produced by reaction between the pickle liquor and the leach liquor can be added to the steel melt directly or reduced to ferro-molybdenum in an electric furnace.

Conclusions

Molybdenum sulfide was found to leach readily in KOH solutions at relatively moderate oxygen pressure and temperature. The observed kinetics of the reaction can be explained by a steady-state analysis involving adsorption of oxygen at the surface followed by a configurational rearrangement of the oxygen on the surface. The oxidation product is then removed from the surface by a reaction with hydroxyl ion forming a water-soluble molybdate. The hydroxyl ion dependency is believed to be diffusion-controlled. A secondary hydroxyl ion dependency is due to a decrease in the solubility of oxygen in increasingly concentrated solutions of KOH. Heat of activation for adsorption of oxygen was found to be 11.84 kcal per mol. Heat of activation for the configuration change was found to be 6.60 kcal per mol. The small apparent number of active sites present suggests that the reaction occurs predominantly at the edges of the molybdenite platelets.

Although the process of high temperature leaching of molybdenite and some of its suggested applications may not be economically feasible at the present state of technology, it is felt that further application of such systems will produce a technique that will allow a process of this type to compete favorably with existing processes.

Acknowledgments

The authors express their appreciation to the Atomic Energy Commission for the funds that made this work possible and to the University of Utah Research Fund for funds for the equipment. Thanks are also extended to John R. Lewis, head of the Dept. of Metallurgy, for his support and interest in the work and to Noble M. Nerheim, graduate student, for work on the bench-scale studies.

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Discussion of this paper sent (2 copies) to AIME by Sept. 30, 1956, will appear in AIME *Transactions* Vols. 205 and 209 and in MINING ENGINEERING and JOURNAL OF METALS, May 1957.

The work presented here was sponsored by the Atomic Energy Commission under Contract No. AT (11-1)-82, Project No. 1.

This paper represents a portion of the thesis to be submitted by William H. Dresher in partial fulfillment of the requirements for the degree of Doctor of Philosophy at the University of Utah.

aime news

Mining Branch Lists 1957 Nominations

This is the complete list of officers nominated for the Mining Branch and its Divisions. Nominations of AIME officers for 1957 appear on pages 742 to 743.

Minerals Beneficiation Div.

Chairman, William B. Stephenson, Allan-Sherman-Hoff Pump Co.; Associate Chairman, Nathaniel Arbiter, School of Mines, Columbia University; Vice Chairman, Wayne L. Dowdey, Eimco Corp.; and John C. Lokken, Dow Chemical Co.; Secretary-Treasurer, H. Rush Spedden, Electro Metallurgical Co.; Executive Committee: Above officers; and Past Chairman Norman Weiss, American Smelting & Refining Co.

Mining, Geology, and Geophysics Div.

Chairman, Elmer Jones, St. Joseph Lead Co.; Mining Subdivision Vice Chairman, Clark Wilson, New Park Mining Co.; Geology Subdivision Vice Chairman, Ed Wisser, University of California; Geophysics Subdivision Vice Chairman, Ralph Holmer, Bear Creek Mining Co.

Industrial Minerals Div.

Chairman, T. L. Kesler, Foote Mineral Co.; Northeastern Vice Chairman, R. M. Grogan, E. I. du Pont de Nemours & Co.; Southeastern Vice Chairman, R. H. Feierabend, Freeport Sulphur Co.; Mid-Continent Vice Chairman, C. F. Deiss, Indiana University; Rocky Mountain Vice Chairman, J. J. Norton, U. S. Geological Survey; Southwestern Vice Chairman, V. E. Scheid, University of Nevada; Northwestern Vice Chairman, R. S. Mason, Oregon State Dept. of Geological & Mineral Industries; Canadian Vice Chairman, H. M. Woodrooffe, Mines Bureau, Ottawa, Canada; Secretary-Treasurer, J. G. Broughton, N. Y. State Science Service; Executive Committee (1957-1960): M. L. Leonardi, American Potash & Chemical Corp.; J. B. Graham, Leggette, Brashears & Graham; C. F. Clausen, Portland Cement Assn.; (1957-1958): A. L. Hall, Cabot Carbon Co.; W. F. Rappold, General Refractories Co.

Minerals Economics Div.

Chairman, Louis C. Raymond, Ford, Bacon & Davis Inc.; Vice Chairman, Lorin A. Cranson, Honolulu Oil Corp.; Henry T. Mudd, Cyprus Mines Corp.; Adam K. Stricker, Jr., General Motors Corp.; Secretary-Treasurer, Paul F. Yopes, U. S. Bureau of Mines; Executive Committee, (1957-1960): W. F. Dietrich, USBM; J. K. Richardson, Kennecott Copper Corp.; Sheldon P. Wimpfen, Glen Alden Coal Corp.

Coal Division

Chairman, C. T. Holland, Virginia Polytechnic Institute; Chairman-elect, J. W. Woomer, J. W. Woomer & Assoc.; Secretary-Treasurer, David R. Mitchell, Pennsylvania State College; Executive Committee (1957-1960): C. J. Potter, Rochester & Pittsburgh Coal Co.; James A. Hagy, Jewell Ridge Coal Corp.; W. L. Wearly, Joy Mfg. Co.

Mineral Industry Education Div.

Chairman, John C. Calhoun, Texas A&M; Vice Chairman, O. Cutler Shepard, Stanford University; Secretary-Treasurer . . . . (To be elected Feb. 1957); Executive Committee (1957-1960): Mining, Koehler S. Stout, Montana School of Mines; Metals, A. W. Schlechten, Missouri School of Mines; Petroleum, B. C. Craft, Louisiana State University.

Fuels Conference Set for October in Capitol

Washington, D. C., will be host city for the 19th Annual National Solid Fuels Conference October 25-26, 1956. Members of two sponsoring engineering societies, ASME and AIME and representatives of solid fuels producers and consumers, will assemble at the Sheraton Park Hotel for a two-day discussion of technical and economic developments of importance to the national economy. Theme of the conference will be *The Future Role of Solid Fuels in an Expanding Economy.*

Final arrangements for the conference are being completed by local members of ASME and AIME. The local committee includes Leroy F. Deming, chairman; Louis C. McCabe, co-chairman; C. A. Reed; C. F. Hardy; William L. Crentz; R. E. Jack; William Bradbury and Mrs. A. Wiley Sherwood. Mr. Deming, general chairman of the conference, is head of the power generation section, Bureau of Yards and Docks, U. S. Navy, Washington, D. C. He is serving in place of Louis R. Caplan, incapacitated by illness.

More than 400 engineers, guests and ladies are expected to attend. Special sight-seeing tours and social events have been arranged for the ladies. Additional attractions are being planned for those out-of-town visitors who may wish to extend their stay through the week-end following the conference.

Results of AIME Membership Survey

A review of the membership applications of 296 members of full professional grade selected at random, (slightly over 2 pct of AIME members) showed that 224 or 75.7 pct, had a college degree; 52, or 17.6 pct, attended college but did not graduate; and 20, or 6.7 pct never attended college.

Of 113 Associate Members, (slightly over 5 pct of all AIME members in that grade), 75, or 66.3 pct, had a college degree; 25, or 22.1 pct, went to college but received no degree; and 13, or 11.5 pct did not attend college. Of 153 applications received from Junior Members, (2.8 pct of the total number) only one did not go to college.

New Douglas Medal Committee Chairman

Carleton C. Long has been named new chairman of the Douglas Medal Committee. He will serve in place of Walter A. Dean, who was unable to continue in this capacity, due to the pressure of his own work.

1957 AIME Nominations Slate Announced

For President-Elect

AUGUSTUS B. KINZEL

Augustus B. Kinzel has been nominated for the presidency of AIME in 1957. Vice president of research, Union Carbide & Carbon Corp., N. Y., Dr. Kinzel is a native of New York City and studied at Columbia University, MIT, and the University of Nancy, France, where he received his Doctorate in 1922. He has been AEC chief consultant in metallurgy since 1943, and is a member of the advisory panel on general sciences, Dept. of Defense. Dr. Kinzel, who joined AIME in 1926, has served the Institute as director and was vice president in 1955; he was chosen Howe Memorial Lecturer for 1952. Dr. Kinzel began his career as a metallurgist for General Electric Co., Pittsfield, Mass. in 1919, and joined Union Carbide's research department in 1926. The candidate is a member of the following organizations: ASM, ASTM, SAE, AISI, International Acetylene Assn., and the International Institute of Welding. He has contributed numerous articles to technical publications and is also the author of *Alloys of Iron and Chromium*, volumes 1 and 2.

For Vice Presidents

E. C. Babson, manager, Canadian Div., Union Oil Co. of California, in Calgary, Alberta since 1951, has been nominated as AIME vice president. A native of Los Angeles, he attended grade and high school in Portland, Ore., and then went on to study metallurgy at Stanford University where he received his A.B. in 1930 and M.E. in 1933. Mr. Babson began his career with Union Oil, as a research chemist in Wilmington, Calif., later becoming production engineer. For two years he lectured in metallurgy as part of the war training program at the University of Southern California. Mr. Babson was a partner of Peerless Pacific Co., Portland, until 1948 and then returned to Union Oil in Calif. as staff engineer. A year later he was promoted to assistant chief petroleum engineer. He has been chairman of the Pacific Coast District, American Petroleum Institute, and acting chief, Reserves Section, Production Div., Petroleum Administration for Defense. Mr. Babson is serving as AIME Director through 1957.

E. C. BABSON

Roger V. Pierce, a member of AIME since 1935, has been nominated as director for a three-year term. Born in Pacific City, Wash., he attended the Montana School of Mines in Butte, where he obtained his B.S. and M.S. degrees. Mr. Pierce was first employed as miner and shift boss at The Anaconda Co. in 1936. A year later he joined Ingersoll Rand Co. where he worked for ten years as mechanization specialist for Canada, Mexico and the U. S. He has been consultant to such companies as Cerro de Pasco, and San Francisco Chemical Co. Since 1950, Mr. Pierce has been consulting mining engineer and president of Machinery Center Inc. Active in AIME, he was chairman, section delegate and a member of the executive committee of the Utah Section. In addition, Mr. Pierce has headed the technical publications committee, Mining Subdivision, which he also served as program chairman.

R. V. PIERCE

For Directors

Luther C. Campbell, a nominee for the 1957 Board of Directors, is a vice-president of the coal division, Eastern Gas & Fuel Assoc., Pittsburgh, and has served as president of the National Coal Assn. since 1952. Born in Renovo, Pa., he graduated from Pennsylvania State University in 1915 with a degree in mining engineering. A mining official since 1915, first with Rand Mining Co., Nev., he was later general superintendent of coal mining operations for McKinney Steel Co. in Ky. In 1927 he held this position at Melcroft Co., then at Elkhorn Piney Coal Mining Co.; both companies are now part of Eastern Gas & Fuel Assoc., of which he has been vice president since 1943. Mr. Campbell joined AIME in 1935, and from 1946-1948 was a member of its committee for promotion of student interest in coal

John S. Bell, Chairman of the Nominating Committee, has announced nominations for President-Elect, Vice Presidents, and Directors for 1957. As provided in Art. IX, Sec. 2, of the bylaws, 25 Members or Associate Members may sign and transmit to the Secretary's office prior to September 1, 1956 "any complete or partial ticket of nominees," should they wish other candidates to be considered.

In such instance, a letter ballot will be forwarded to all Members in good standing in the United States, Canada, and Mexico, tabulating both the official ballot and any supplementary nominations. If no supplementary nominations are thus received, no letter ballot will be printed, and nominees on the official ballot shall be declared duly elected at the meeting of the Board of Directors or the Executive Committee in November.

mining as a career. He served as chairman of the NCA safety committee and organized their safety division in 1948. For a number of years he has participated actively in the work of Bituminous Coal Research Inc., the American Mining Congress, and the National Safety Council, as well as 11 other regional engineering and mining organizations. Since 1945 Mr. Campbell has been a member of the Old Timers Club, a group of industry leaders dedicated to helping young men attain professional careers in the coal industry.

L. C.
CAMPBELL

Thomas C. Frick, chairman of AIME Petroleum Branch 1956, has been nominated director for a three-year term. Past-chairman of East Texas Local Section and of the Permian Basin Section, he is a native of Collinsville, Oklahoma and attended Missouri School of Mines. He transferred to the University of Tulsa, graduating in 1933, with the first class in petroleum engineering. Mr. Frick joined the refining department of Phillips Petroleum Co. where he worked for three years as engineer. He joined AIME in 1936 shortly before returning to the University of Tulsa where he was assistant professor and head of the production department of the School of Petroleum Engineering. In 1941 Mr. Frick accepted a position with Atlantic Refining Co. in Dallas as job analyst, and in 1944 returned to petroleum engineering as district superintendent for the company in East Texas; he was regional manager for Atlantic from 1951-1954 and last year assumed his present post, as manager, Production Div. of the Domestic Crude Oil Production Dept.

T. C.
FRICK

F. W.
STRANDBERG

Fred W. Strandberg, recently promoted to chief mining engineer at The Anaconda Copper Mining Co., Butte, Mont., has been nominated as an AIME director for a three-year term. Born and raised in Iowa, he joined Anaconda in 1916 after graduating from Iowa State College with a B.S. degree in mining engineering. He has been with that company ever since, except for a brief stint at the Walker Mine in California from 1930-1933. A member of AIME since 1939, Mr. Strandberg was Montana section delegate from 1950-1955, and chairman of the section in 1953.

Arthur W. Thornton, assistant to the vice president-operations, National Tube Div., U. S. Steel Corp., Pittsburgh, has been nominated to serve as an AIME director for a three-year term. A member of the Institute since 1947, he has served on the executive committee of the NOHC,

A. W.
THORNTON

the Bessemer Committee, and was ISD chairman. He is chairman-elect of the Metals Branch. Born in McKeesport, Pa., Mr. Thornton graduated from Lehigh University in 1931 with a degree in metallurgical engineering. He was affiliated with Weirton Steel Co. in W. Va., and was a member of the metallurgical department of Duquesne Works of U. S. Steel. Since 1934 he has been with the National Tube Div., U. S. Steel, beginning as steel works metallurgist, then superintendent of the Open Hearth and Bessemer Dept., later assistant general superintendent, attaining his current post in 1954.

James S. Vanick has been nominated as director for a three-year term. His lengthy association with the Institute began in 1921; he has since served AIME as chairman of the New York Section. A research metallurgist for International Nickel Co., N. Y., since 1922, Mr. Vanick is the recipient of the Wm. H. McFadden Gold Medal for 1956, of the American Foundrymen's Society.

J. S.
VANICK

Born in Cleveland, he received his basic engineering training there, at Case Institute of Technology and continued at George Washington University where he obtained his Master's Degree. Mr. Vanick is a member of AFS, ASME, ASM, ASTM, Corrosion Engineers, and the British Iron & Steel Institute, serving these groups as chairman of various committees. He has written over 50 papers for technical meetings and for the trade press, on ordnance materials, cast metals production and corrosion effects.

Lamar Weaver, general superintendent, Tennessee Copper Co., has been nominated an AIME director for a three-year term. Born in Palmetto, Georgia, Mr. Weaver studied at N. Georgia Agricultural College, where he received his B.S in mining engineering. In 1922 he joined Tennessee Copper in Copperhill, Tenn. as a mine engineer, and six years later became assistant superintendent of mines. By 1938 he was superintendent, and in 1953 assumed his present post. A member of AIME since 1938, Mr. Weaver was chairman of the Southeastern Section in 1952 and has been section delegate since 1953.

LAMAR
WEAVER

Dates Announced for Future Board of Directors Meetings

Future meetings of the Board of Directors or the Executive Committee will not be held monthly, as they have been in the past. Election of new members and other matters which require prompt action will be conducted by mail. The dates decided upon for Board meetings were: Nov. 21, 1956, in New York; two meetings in New Orleans are scheduled for Feb. 24 and Feb. 26, 1957.

The Executive and Finance Committees will meet jointly with the Treasurer and Secretary of the Institute in New York on the following dates: Sept. 20 and Dec. 19, 1956.

At the first meeting of the new Board of Directors, Feb. 21, 1956, the following were named for the offices and committees indicated: Gail F. Moulton to succeed himself as Treasurer and Controller for a one-year term; William J. Masson to continue as Assistant Controller; Ernest Kirkendall to succeed himself as Secretary for one year; Edward H. Robie reappointed as Secretary Emeritus, to serve until November 30, 1956.

The Executive Committee is headed by C. E. Reistle, Jr., as chairman; H. DeWitt Smith, vice chairman, and T. B. Counselman, Grover J. Holt, and A. W. Thornton. The Finance Committee will be led by Andrew Fletcher, chairman; R. B. Caples, and C. R. Dodson.

Chairman of the various fund committees were named as follows: Endowment, W. W. Mein, Jr.; Endowment Fund *X*, G. F. Moulton; Rocky Mountain Income, Henry Krumb; Seeley W. Mudd Memorial Fund, John M. Lovejoy; Gemmell Memorial Fund, Louis S. Cates; Henry L. Doherty Memorial, John M. Lovejoy.

Chairmen of the Award Committees are: Robert H. Richards Award, C. Y. Garber; Rand Foundation Award, M. L. Haider; Douglas Metallurgical Medal, C. C. Long; Saunders Mining Medal, C. E. Weed; Lucas Petroleum Medal, R. W. French; Erskine Ramsay Coal Medal, L. C. Campbell; Student Prize Paper Awards, David H. Beetem; Benjamin F. Fairless Award, J. S. Marsh.

AIME Committees will be guided by the following chairmen: Honorary Memberships, Andrew Fletcher; Admissions, R. B. Caples (chairman), and F. A. Ayer (vice chairman); Members' Conduct, W. M. Peirce; AIME Pension Trust, G. F. Moulton.

Chairmen of the AIME intersociety committees are: Library, Frank T. Sisco; Cooperation with the CIM and MSNS, P. J. Shenon; Research, F. B. Foley.

Incentive Offered For Obtaining New Members

An added incentive for securing new members was authorized by the executive committee at their meeting on April 19. The names of all AIME members who obtain five or more new Institute members, will be placed in a hat and five names drawn. The first will receive five years of paid-up AIME dues and the other four, one year of paid-up dues each. This policy has been in effect in the Petroleum Branch and is now extended to the other two branches as well. Until now, AIME has presented a blue and white membership award pin to any member instrumental in securing five new recruits.

Will Publicize AIME in 1956

Charles Bayer, who successfully handled publicity and news releases for the Annual Meeting last February, has been retained by AIME for the rest of the year. It is planned to issue suitable releases on important AIME meetings, publications, appointments, and awards.

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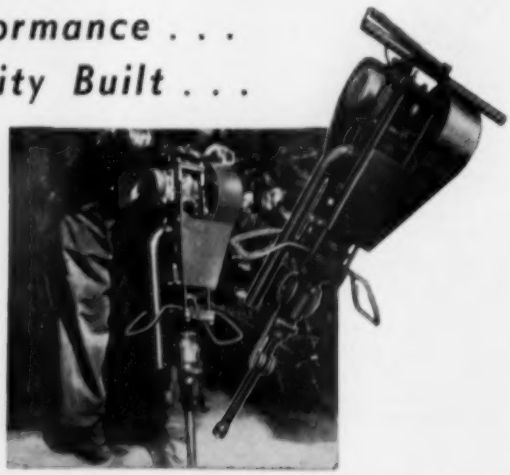

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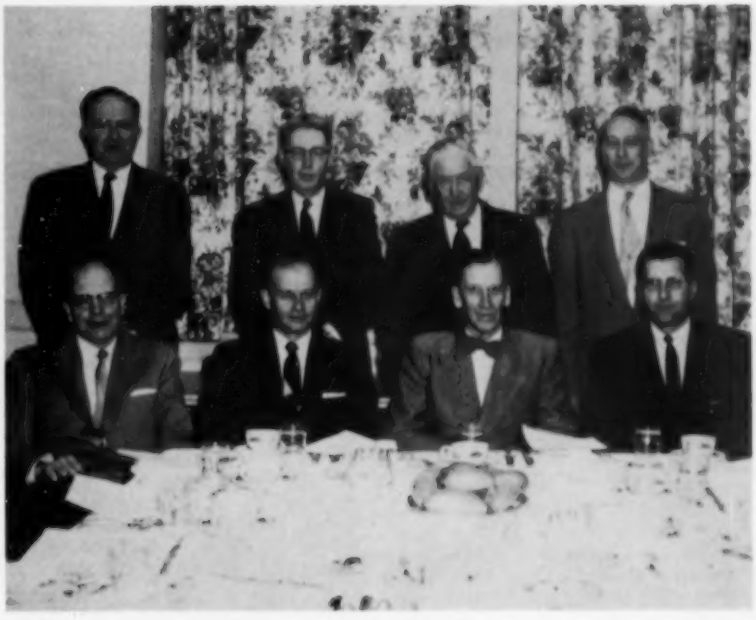
Around the Sections

AIME President Tours Western Local Sections

Mr. Reistle made a tour of some of the Local Sections in April. His first stop was at Salt Lake City, where he was the guest of honor at the annual spring dinner-dance of the Utah Section. Mr. Reistle then went on to Butte and attended a luncheon and a banquet given for him by the Montana Section at which he was the principal speaker. He also spoke at a convocation of students and faculty at the Montana School of Mines. The next day Mr. Reistle left for Spokane and attended a meeting of the Columbia Section. On April 11 he was the guest of honor at a dinner party held by the Coeur d'Alene Subsection at the Union Legion Hall, Kellogg, Idaho. Mr. Reistle also visited the San Francisco, Nevada, and Chicago Sections. He was accompanied on his tour by Mrs. Reistle, Mr. Kirkendall, and Mr. O'Brien.



This photograph was taken at the luncheon held on the occasion of the visit by Carl E. Reistle, Jr., AIME President, to Butte. Left to right: Seated—J. R. Van Pelt, president, Montana School of Mines, and Montana Section Chairman; Mr. Reistle; F. W. Strandberg, chief engineer, The Anaconda Co.; and C. J. Hicks, Montana Section Secretary. Standing—E. I. Renouard, Jr., manager of mines, The Anaconda Co.; Roy E. O'Brien, AIME Field Secretary; E. P. Shea, head, Geological Dept., The Anaconda Co.; and Ernest Kirkendall, AIME Secretary.

Pictured above are some of the guests and sponsors of the banquet held in Constitutional Hall, University of Alaska School of Mines.

• The **Alaska Section** and the **Woman's Auxiliary** of that section were co-sponsors of a banquet held in April in Constitution Hall on the campus of the University of Alaska School of Mines. Graduating seniors from the university with their escorts were the guests of honor. Other guests from the university were: Ernest N. Patty, president, and Mrs. Patty; Neil W. Hosley, dean, and Mrs. Hosley; and students of the School of Mines. The toastmaster was Theodore Loftus, vice president of the Alaska Section. Earl H. Beistline, dean, School of Mines, introduced the seniors and James D. Crawford, counselor to the Mining Society on the campus, presented them with scrolls containing the Canons of Ethics for Engineers. The scrolls were donated by the AIME. Mrs. Donald MacDonald III, chairman of the Woman's Auxiliary of the Alaska Section, presented the seniors' wives and escorts with corsages provided by the WAAIME. A highlight of the evening was the presentation of a Brunton and accessories to Pedro Denton, an outstanding senior. Speakers at the banquet were Anson LeVake Renshaw, Jr., president of the Mining Society, Mrs. MacDonald, Earl H. Beistline, Douglas Colp, and Ernest N. Patty. Later in the evening both the Alaska Section and the Woman's Auxiliary held short business meetings. Members of the committee for arrangements were: Bruce Thomas, Maurice Butler, Theodore Loftus, and Donald MacDonald III.

• Grover J. Holt was the guest of honor at the annual dinner-dance held by the **St. Louis Section** on May 11 at the Missouri Athletic Club.

• The **El Paso Metals Section** and the Kidd Mining Club met on April 19 at Seamon Hall, Texas Western College. After a bean feed senior students presented papers on their research problems.

• The **Washington, D. C., Section** held a dinner-meeting on May 8 at the Cosmos Club. The principal speaker was Simon D. Strauss, vice president, American Smelting & Refining Corp. Mr. Strauss, who had recently returned from a trip to Europe, discussed the state of the mineral industries on that Continent.

Educators Meet With National Coal Assn.

The Vocational Training and Education Committee of National Coal Assn. and the Advisory Committee of the department of mining and metallurgical engineering held a combined meeting at the University of Illinois on April 20-21, 1956.

The Friday morning schedule included a tour of the laboratories to see facilities in the Illinois Geological Survey. At noon, Dean William L. Everitt of the college of engineering, spoke on the continuity of education from grade school through college and during employment after graduation.

The afternoon was devoted to discussion of various fields: F. D. Wright described courses directly associated with mining; T. A. Read reported on metallurgical instruction for mining students; geology was discussed by George W. White; E. C. Jordan told about the electrical courses; and theoretical and applied mechanics offered to the students in the Mining Dept. were described by T. J. Dolan. A report on coal research was then presented by J. A. Simon of the coal division of the Geological Survey.

Friday evening, Henning Larsen, Provost of the University of Illinois, talked about outstanding members of the faculty and work they had done to advance knowledge in their respective areas. He also referred to the valuable facilities at the university. Dr. Larsen was introduced by Henry C. Woods, chairman of both committees.

A new scholarship in fuel technology, and mineral engineering fellowships at Pennsylvania State University were discussed at the executive session on Saturday. Reference was made to the short course in coal preparation offered by West Virginia University Ext. Dept.; the University-Industry Cooperative Program in Mining at the University of Pittsburgh. A University of Illinois booklet entitled *Careers in Mining Engineering* was mentioned and its distribution to companies and educators recommended. Scholarships recently announced by Pocahontas Fuel Co. Inc., were accepted by members of the committees as evidence of coal companies' desire to aid students interested in mining engineering.

The use of coal in heating buildings was treated by J. R. Carroll, department of mechanical engineering, who explained how engineers cooperate with architects in plans for heating and ventilating. The conclusion of a discussion on employment was that industry could use two technicians for each graduate engineer. Recent employment activities showed that the mining industry had difficulty in meeting the offers made by some industries. C. G. Evans reported on extension courses taken by employees of his company who were encouraged to study administration, accounting and combustion, with the understanding that the company would pay one-half the cost for a grade of C, and the entire cost for those who obtained higher grades. M. E. Speare gave an interesting report on the distribution of information on coal to schools, editing of text books, and work of the Speakers' Bureau.

The fluorspar industry was described by Gill Montgomery, a member of the Advisory Committee. He commented on methods of mining, costs and selling prices, difficulty of competing with foreign sources, and use of older engineers when younger men could not be obtained.

The meeting adjourned with the agreement to accept the invitation of Dean Curtis L. Wilson to visit Missouri School of Mines and Metallurgy in October.

Julius B. Haffner, center, president, International Metals & Oil Corp., received an honorary Doctor of Science degree from the University of Idaho on June 10. It was conferred on him by University President Theophilus, right, for outstanding contributions to Idaho and to the minerals industry. At left, J. D. Forrester, Dean of the College of Mines.

AIME Awards Funds

Funds are sufficient to finance all of the usual AIME Awards to be presented at the Annual Meeting in New Orleans February 24 to 28, 1957, except the Ramsay Medal for achievement in coal mining.

Award committees have been named, and the chairmen will be glad to hear from any AIME members who wish to suggest the name of a candidate worthy for consideration by the appropriate committee. Their names follow: C. C. Long, committee chairman, *James Douglas Medal* for distinguished achievement in nonferrous metallurgy; C. E. Weed, chairman, *W. L. Saunders Medal* for mining; R. W. French, chairman, *Anthony F. Lucas Medal* for petroleum; M. L. Haider, chairman, *Charles F. Rand Medal* for mining administration; C. Y. Garber, chairman, *Robert H. Richards Award* for mineral beneficiation; J. S. Marsh, chairman, *Benjamin F. Fairless Award* for iron and steel; C. M. Squarcy, chairman, *J. E. Johnson Jr., Award* for iron and steel (recipients under 40); J. C. Calhoun Jr., division chairman, *Mineral Industry Education Award;* K. L. Cook, division chairman, *Jackling Award* for mining, geology, or geophysics.

The following awards have also been authorized for best papers: *Robert W. Hunt Medal, Mathewson Medal, Extractive Metallurgy Division Award,* and the *Rossiter W. Raymond Award.*

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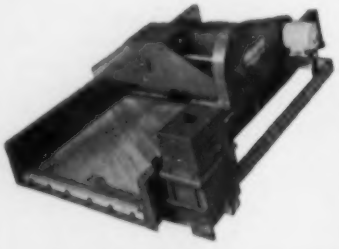

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PERSONALS

JOHN P. HAMM

John P. Hamm is assistant to the president, Western Machinery Co., San Francisco, Calif. Mr. Hamm has been associated with the company in sales and administrative functions since 1951. After graduating from the University of Colorado in 1934, he was engaged in Colorado mining ventures. Mr. Hamm is a past president of the Colorado Mining Assn.

John H. Baker is assistant to the general manager of production, American Gilsonite Co., Salt Lake City. Mr. Baker joined the company as a mining engineer at Bonanza, Utah, and was promoted to mine superintendent in 1949. He is succeeded at Bonanza by **Paul Borden.**

Cloyd M. Smith, consulting engineer, has been appointed Washington representative for World Mining Consultants Inc. Mr. Smith will continue to serve his former clients.

H. R. BURCH

H. R. Burch has been named assistant general manager, Weed Heights, Nev., operations, The Anaconda Co. **C. J. Houck** succeeds Mr. Burch in the position of mine superintendent, and **D. K. Gill** succeeds Mr. Houck as general mine foreman.

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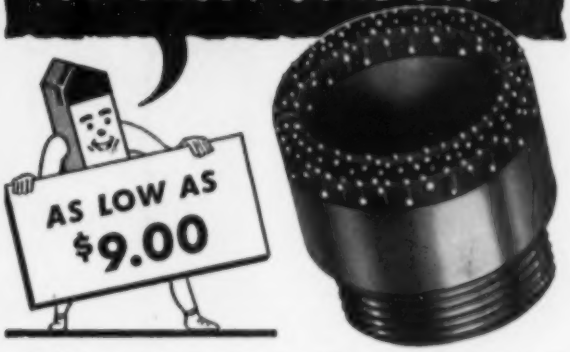

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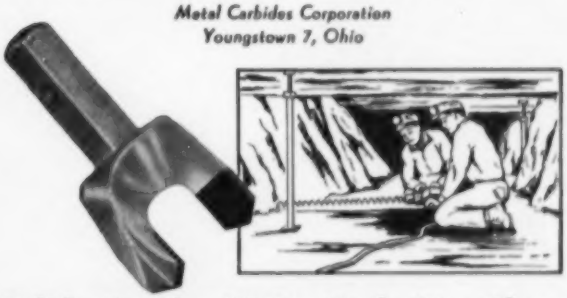

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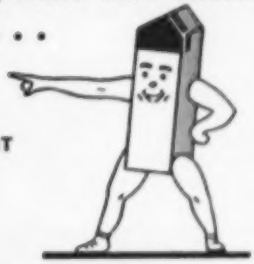

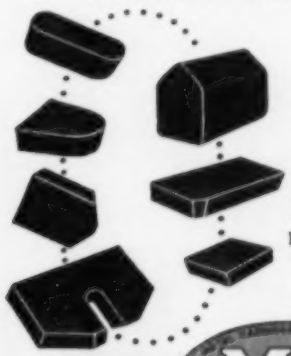

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Clayton A. Carpenter, formerly section leader, Research Center, Union Oil Co. of California, Brea, Calif., has returned to Nevada after an absence of 15 years. Mr. Carpenter is working on his own ranch near Yerington, Nev., specializing in hay and grain.

J. W. Snavely has been named assistant manager, Conveyor and Process Equipment Div., Chain Belt Co., Milwaukee. He will continue as manager of conveyor sales. Mr. Snavely has been with Chain Belt Co. since 1927.

A. L. Hayes has been appointed assistant to the vice president responsible for mining and exploration, The New Jersey Zinc Co., with headquarters in New York. Mr. Hayes joined the company in 1927 as an engineer at Gilman, Colo. He has been assistant to the manager of mines since 1950.

William Bellano has been promoted to the newly created position of director, mining and minerals exploration, International Minerals & Chemical Corp. He was formerly production manager for the company's phosphate chemicals division, in charge of the Bonnie plant. Mr. Bellano's headquarters will be in Chicago.

Paul Gemmill was named general manager of Combined Metals Corp.'s Utah and Nevada operations. He was formerly a mining engineer for the company.

Grover C. Dillman, will retire as president of Michigan College of Mining and Technology, on Oct. 1, 1956. Under his administration, which began in 1935, the enrollment has increased fourfold and the college had its greatest expansion in curricula, new facilities and buildings. In public life for 41 years, Mr. Dillman joined the staff of Michigan State College in 1913 and was State Highway Commissioner from 1929 to 1933. He also served as director of the Welfare Dept., director of the state budget, and president of the Michigan Engineering Society.

ARTHUR C. DAMAN

Arthur C. Daman, president, Denver Equipment Co., Denver; **Franklin Coolbaugh,** vice-president, Climax Molybdenum Co., Climax, Colo; **Philip J. McGuire,** Dorr-Oliver Inc., Oakland, Calif.; **Charles Francis Jordan, Jr.,** of El Paso, Texas; **Russell H. Volk,** president, Plains Exploration Co.; and **Philip M. McHugh** (posthumously); were awarded distinguished achievement medals from the Colorado School of Mines, during the school's 82nd commencement exercises in May. These medals are given to School of Mines alumni for outstanding service and are roughly equivalent to an honorary degree.

William L. Zeller, formerly senior engineer, Warner Collieries, Co., Mammoth, W. Va., is now a partner in the firm of Hurley & Zeller, R. D. 2, Ligonier, Pa.

William F. Rappold, district sales manager, General Refractories Co. in Salt Lake City, was elected chairman of the Natural Resources Commission, Utah Chamber of Commerce Executives' Assn. Mr. Rappold succeeds **Clark L. Wilson** of New Park Mining Co.

George F. Sipp is assistant chief construction supervisor, Raymond Concrete Pile Co. of Delaware, working in Ankara, Turkey.

Herbert M. Weisz has been appointed general manager, Cía. Minera de Bolivia, Empresa Minera Catavi, Bolivia.

A. G. Mosier is assistant mill metallurgist, Braden Copper Co., Rancagua, Chile.

Jean F. Dardé is now a consulting mining engineer Granja La Guadalupana, Los Gavilanes, Santa Anita, Jalisco, Mexico. Mr. Dardé was unit superintendent, Cía. Minera de Penoles, Sinaloa, Mexico.

Albert E. Forster, president, Hercules Powder Co., Wilmington, Del., since 1953, has been elected chairman of the board. Mr. Forster succeeds **Anson B. Nixon** who has retired. Mr. Nixon will continue as a director.

L. A. Dahners is mine superintendent, Lepanto Consolidated Mining Co., Mountain Province, P. I. Prior to joining Lepanto Mr. Dahners was with Cyprus Mines Corp., Cyprus, for more than six years.

Michael Burke has left Gardner-Denver Co. and joined the staff of Minas de Acebo, S. A. de C. V., Hermosillo, Sonora, Mexico.

H. A. Trestrail is with Straits Consolidated Tin Mines Ltd., Kroh, Upper Perak, Malaya.

C. O. Perlwitz has resigned as construction engineer for Northern Peru Mining Corp., Trujillo, Peru, and is now in the U. S.

Edward W. Bullard, San Francisco Advisory Committee, Engineering Societies Personnel Service Inc., has resigned. Mr. Bullard, president of the E. D. Bullard Co., represented his Section on the Committee for 18 years, the last seven years as Chairman. He was presented with a certificate for outstanding service to the engineering profession from the ESPS National Board of Directors. **Herbert A. Sawin,** assistant sales director, Yuba Mfg. Co., has accepted appointment by **James P. Bradley,** Chairman of the San Francisco Section AIME, as representative to the San Francisco Advisory Committee for the ESPS. Mr. Sawin and representatives from other national engineering societies will counsel, guide, and review ESPS operations to assure satisfactory employment service to Western engineers.

J. Reysenbach has joined Uis Tin Mining Co. Ltd., Uis, S. W. Africa.

Frank L. Beggs, formerly mining and quarry superintendent, Coplay Cement Mfg. Co., Coplay, Pa., has been appointed production manager, Carbola Chemical Co. Inc., Natural Bridge, N. Y.

Floyd W. Erickson has been appointed to the new position of manager, Babbitt Div., Reserve Mining Co. Mr. Erickson joined the com-

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pany in 1951 as chief mining engineer at Babbitt. Since last May he has been general superintendent at Silver Bay. Prior to joining Reserve Mining Co. he spent nine years with tin and copper mining companies in Chile and Bolivia, and was employed by the U. S. Engineers and the U. S. Bureau of Mines.

F. V. C. Hewett, consulting mining engineer, is now president and managing director, McIntyre Porcupine Mines Ltd., Toronto.

Charles R. Pace, Jr., has joined Rockville Crushed Stone Co., Rockville, Md. Mr. Pace was with Manganese Mining & Contracting Corp., Marion, Va.

Frank L. Pehrson is senior research scientist, Kennecott Research Center, Salt Lake City. Mr. Pehrson was metallurgical and experimental engineer at the Arthur plant, Kennecott Copper Co., Garfield, Utah.

W. H. LESSER

W. H. Lesser of Scranton, Pa., was chosen *Engineer of the Year* by the Northeast Chapter, Pennsylvania Society of Professional Engineers, at their annual meeting May 19. The citation covers Mr. Lesser's 50 years in the engineering and management of anthracite mines, his specialties being the mechanical and electrical aspects, and coal beneficiation. A native of the anthracite mining area, he is an honor graduate of Lehigh University. For more than 20 years Mr. Lesser was with Pierce Management Inc. where he participated in the engineering of many domestic and foreign projects. At present he is a consulting mechanical engineer for the USBM, working on the anthracite flood prevention project.

Clyde E. Weed has been elected president, The Anaconda Co. Mr. Weed, who has been vice president in charge of operations since 1952, will also head three Anaconda subsidiaries: Chile Copper Co., Chile Exploration Co., and Andes Copper Mining Co. He is the first engineer to head the company since 1915. Mr. Weed joined Anaconda in 1938 as general manager. He was awarded

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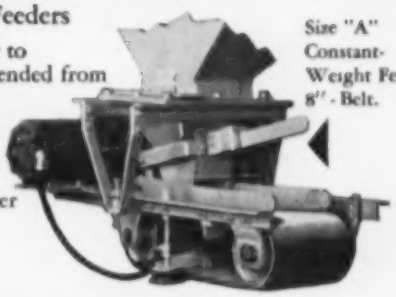
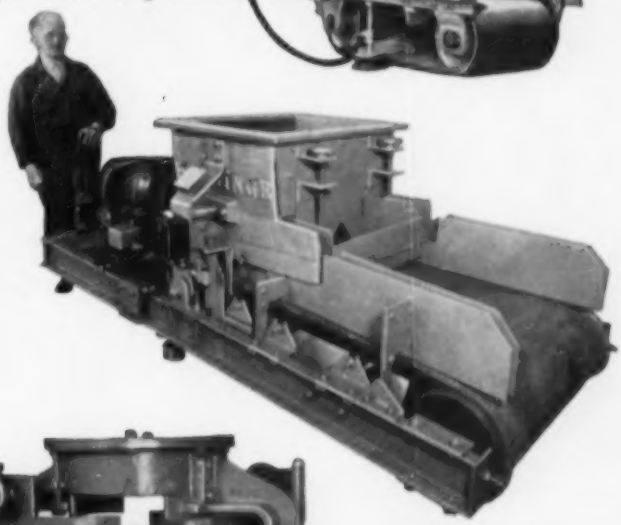
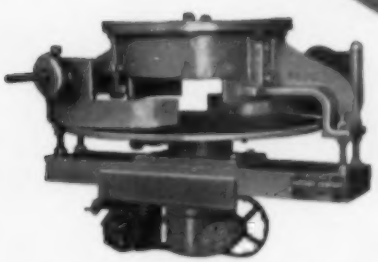


Size "A" Constant-Weight Feeder, 8" - Belt.





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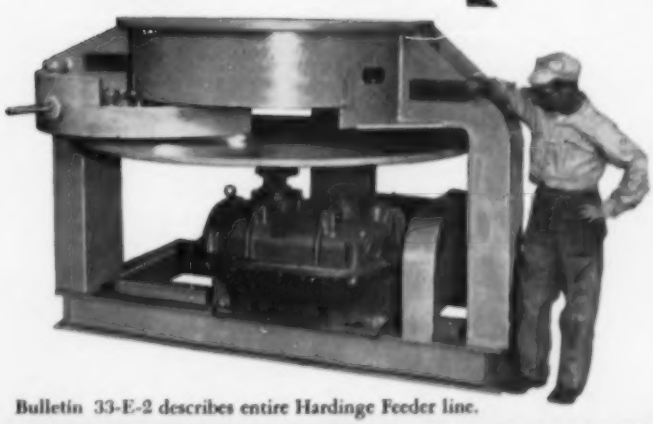

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the AIME William Lawrence Saunders Medal in 1951. Mr. Weed succeeds **Robert E. Dwyer** who has retired after 53 years of service with the company. **Elbert O. Sowerwine,** vice president, has also retired. Mr. Sowerwine joined Anaconda 49 years ago.

Other promotions announced by Anaconda are: **Thomas A. Campbell,** executive vice president, Chile Exploration Co. and Andes Copper Mining Co., to a vice president of Anaconda and president of Anaconda Sales Co.; **Charles Brinckerhoff,** general manager in charge of S. American operations, to vice president, of Chile Exploration Co. and Andes Copper Mining Co.; and **W. Kenneth Daly,** controller since 1940, to a vice president of Anaconda.

W. A. Vine has resigned as associate professor, mining engineering, Missouri School of Mines and Metallurgy, Rolla, Mo. Mr. Vine is now safety and industrial engineer, Consolidated Coppermines Corp., Kimberly, Nev.

Donald M. Ray is mill superintendent, Pima Mining Co., Tucson, Ariz.

Kalman V. Illyefalvi is now associated with Emad Corp., New York, in the purchasing and selling of ores and mineral concentrates.

Archie J. McDonell has been appointed manager of the Sunbright plant, Foote Mineral Co. Mr. McDonell was chief engineer and assistant plant manager, National Lead Co., Tahawus, N. Y. His earlier background includes five years with Braborne Mines Ltd., British Columbia, and three years with Cuyuni Goldfields Ltd., British Guiana. Mr. McDonell graduated from the University of Idaho with a B.S. degree in mining and metallurgy.

LYLE HALLBERG

Lyle Hallberg is mineral testing engineer, Western Machinery Co., San Francisco. Mr. Hallberg was junior metallurgist, American Smelting & Refining Co., Colville, Wash., and has been active in mining operations in Idaho and Montana.

OBITUARIES

Joseph C. Long

An Appreciation by
G. F. Coope, Jr.

Joseph C. Long (Member 1949) died unexpectedly at the age of 43 following a gall bladder operation on Apr. 1, 1956. His premature death cut short a career that had already contributed much to all phases of the phosphate mining industry and had promised to contribute far more in the future. J. C., as he was usually called, was born near Brooksville, Fla. on June 1, 1912. His father died while J. C. was still a baby. He spent his early years in Citrus County. In his late teens he went to live with a sister in Brewster, Fla., which is in the heart of the Florida pebble phosphate district. Financial necessity obliged him to take a full time job at an early age and miss out on a college education. However, he did find time to take extension courses on technical subjects from the University of Florida. He worked for American Cyanamid, starting as a feed tender in the flotation plant and advancing through the ranks until he was appointed superintendent of their Saddle Creek flotation plant. While with American Cyanamid he met Carlotta Ropp, a school teacher at Brewster, and they were married in 1938.

In 1946 he joined the Virginia-Carolina Chemical Corp. which was erecting a new flotation plant at its Homeland mine. Starting from scratch with a new plant and untrained personnel he developed a flotation operation that became a model of efficiency for other operators in the district. Despite his lack of a technical degree J. C. was a student of the literature of flotation and he achieved the status of a real authority on phosphate flotation. He was always glad to share his knowledge and experience with others and there is not an operator in the field who has not availed himself of J. C.'s generosity in this respect.

A tribute to his unselfishness is a list of the organizations in which he was active and to which he devoted much of his spare time for the betterment of the community. He was on the board of stewards of the College Heights Methodist Church and in charge of the Boy Scout Committee of that church as well as treasurer of the Sunday School Class. He was also a 32nd degree Mason and a member of the Scottish Rites and the Eastern Star.

He leaves his wife, Carlotta, and three children, Joseph C., Jr., Julia Margaret, and Mary Lucille. He is also survived by his mother, Mrs. Mary A. Long, and by three sisters, Mrs. E. H. Thompson, Mrs. T. H. Fay, and Mrs. J. C. Purcell.

J. C. Long's professional ability, his community leadership, and above all his warm personality and his quiet sense of humor will be greatly missed by his numerous friends.

Heath McClung Steele

An Appreciation by
Harold K. Hochschild

Heath McClung Steele (Member 1922) died in New York City on Feb. 21 at the age of 71. He had been in poor health for about two years.

Born in Knoxville, Tennessee, in 1884, Mr. Steele was educated at the Baker-Himmel School and the University of Tennessee. After practical mining experience in Tennessee, Virginia, and Nevada, he became associated with J. R. Finlay in 1911 as consulting engineer, with headquarters in New York. In 1917 Mr. Steele joined The American Metal Co. Ltd., and soon afterwards took charge of its Mexican operations. In 1923 he became a director of the company, in New York, and in 1930 was elected vice president. He occupied both offices until his retirement at the end of 1954. Mr. Steele was also president of the Cía. Minera de Penoles, S. A., and Cía. Metalurgia de Penoles, S. A., in Monterrey, Mexico; Southwest Potash Corp., Carlsbad, N. M., and an officer and director of other subsidiaries of The American Metal Co., as well as a director of the Rhodesian enterprises in which the Metal Co. is interested. He was also chairman of the Tsumeb Corp. Ltd., Tsumeb, S. W. Africa, from its organization in 1947 until his retirement in 1954.

During World War II Mr. Steele served as chief of the aluminum-magnesium division, Bauxite-Alu-

(Continued on page 757)

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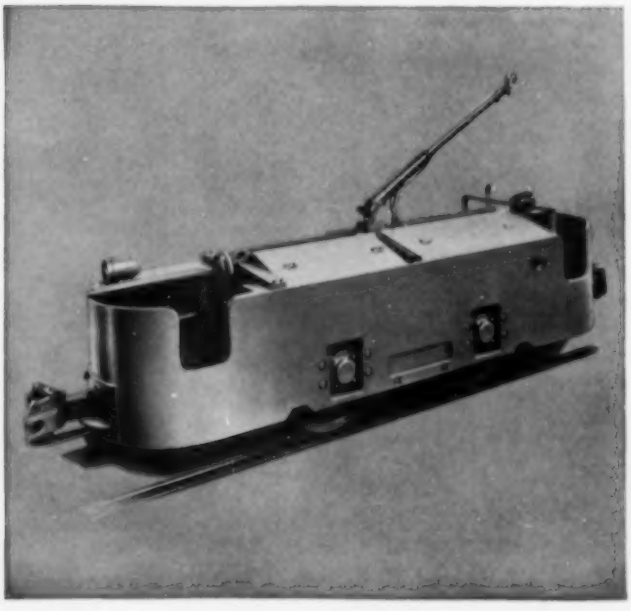

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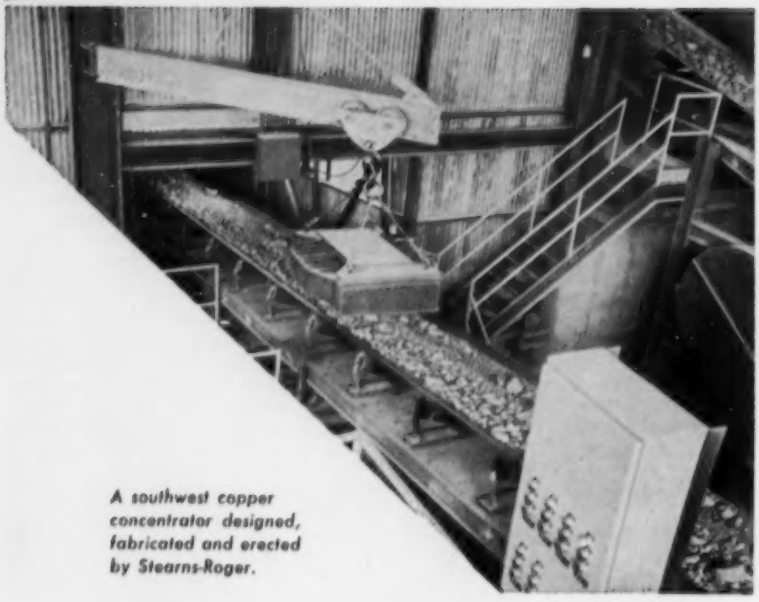

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Total AIME membership on May 31, 1956 was 24,609; in addition 2,685 Student Associates were enrolled.

ADMISSIONS COMMITTEE

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The Institute desires to extend it privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

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John E. Bailey, Monticello, Utah
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Victor C. DeMunck, Butte, Mont.
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Douglas L. Frith, Bethlehem
Elbridge W. Gresseth, Santa Maria, Calif.
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Eugene C. Herschelman, Allentown, Pa.
Paul H. Knowles, Albuquerque, N. M.
Charles J. Kundert, Palo Alto, Calif.
Alan R. Lukens, Newton, Mass.
Clifford A. Mark, Orem, Utah
Richard McCombe, Mahr Tunnel, Peru
Robert N. Morris, Shaker Heights, Ohio
Asher Nesin, Westfield, Mass.
John Prince, Jr., Prairie Village, Kan.
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Raymond P. Maloney, Douglas, Alaska
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REINSTATEMENT

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Edward P. Leach, Bethlehem
Melvine E. William, Santiago, Chile

Student to Member

Charles D. Blancke, Jr., Vinton, La.
Hugo H. Korpinen, Negaunee, Mich.

(Continued from page 755)

mina Branch, Office of Production Management, in Washington.

Mr. Steele's imaginative leadership contributed substantially to the growth of the Metal Co. He played a large part in the evolution of the company's operations in Mexico and of its copper and by-product refinery at Carteret, N. J. The development of the Pecos Mine can be attributed to his initiative. Owned jointly by the Metal Co. and David M. Goodrich and associates, the mine became a large producer of zinc and lead during the 1920's and 1930's. Mr. Steele was primarily responsible for the company's potash enterprise at Carlsbad, N. M., and for the exploration program which led to their discovery in 1954 of the lead-zinc-copper-silver orebodies in New Brunswick, Canada, which have since been named in his honor.

After his retirement Mr. Steele lived at Glen Mary Farms, Great Mills, Md. Surviving are his wife, Florence Truman Steele; his daughter, Mrs. Sheridan Fahnestock; two sons, Heath Warren and David Truman; five grandchildren; and his sister, Mrs. Henry Clay Long.

William J. Felton (Member 1941) died Feb. 10, 1956. Mr. Felton was secretary, The Institution of Mining & Metallurgy, London, England. He was born in London in 1903. After graduating from the School of Economics, University of London, in 1925, Mr. Felton became personal assistant to the director, Gold Coast Geological Survey, West Africa. In 1932 he joined The Institution of Mining & Metallurgy as assistant secretary, becoming secretary in 1939.

Arthur E. Flynn (Member 1919) died Feb. 28, 1956. Mr. Flynn was Dean and Professor of Mining, Nova Scotia Technical College. He was born in London, England, in 1890 and graduated from the Royal School of Mines, London, in 1912. Mr. Flynn was a member and past president of the Canadian Institute of Mining & Metallurgy and of The Mining Society of Nova Scotia.

Walter R. Ingalls (Legion of Honor Member 1891) died Feb. 25, 1956. Mr. Ingalls was a consulting engineer in Georgetown, Mass. He was born in Lynn, Mass. in 1865 and graduated from Massachusetts Institute of Technology in 1886. During the next three years Mr. Ingalls was engaged in the zinc and lead industry in Leadville, Colo. From 1890 to 1892 he was assistant editor, *E&MJ*. Mr. Ingalls then joined Pittsburgh and Mexican Tin Mining Co., Durango, Mexico. In 1893 he opened a practice in New York as consulting mining and metallurgical engineer and visited many mining districts in the U. S., Canada, Belgium, and Poland. Mr. Ingalls rejoined *E&MJ* as editor in 1905, which position he held until 1919 when he returned to private practice as a consulting engineer. He was the author of several technical papers.

Eldred A. Knapp (Member 1923) died Feb. 16, 1956. Mr. Knapp was a director of Knapp & Bates Ltd., London, England. He was born in Camborne, England, in 1893. After graduating from the Camborne School of Mines in 1914 Mr. Knapp worked for several companies including Anglo-Continental Mines Ltd., Nigeria; Dome Mines Ltd., Ontario; and Stearns-Roger Mfg. Co., Denver. Prior to joining Knapp & Bates Ltd. in 1932, he was consulting metallurgist for Burma Corporation Ltd., Burma.

Necrology

Date Elected	Name	Date of Death
1939	L. D. Bartell	Unknown
1946	Robert W. Buzzard	May 3, 1956
1914	Frank T. Donahoe	May 3, 1956
1906	Stanley N. Graham	Feb. 21, 1956
1949	Neal Ham	Mar. 7, 1956
1915	James L. Head	June 3, 1956
1943	John Tate Lewis, Jr.	May 2, 1956
1920	Oliver P. Luetscher	Jan. 3, 1956
1952	J. G. Miller	May 6, 1956
1938	Harry M. Moses	Apr. 1, 1956
1955	P. Ross Pratt	Sept. 15, 1955
1953	Giovanni Puppo	May 30, 1955
1940	R. C. Rutherford	Apr. 25, 1956

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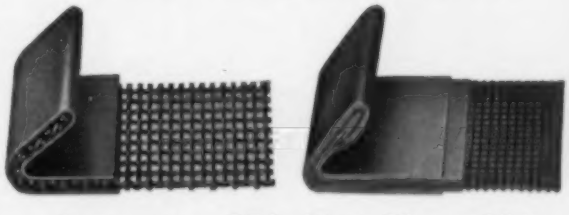

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Coming Events

July 5-7, Mining Soc. of Nova Scotia, annual meeting, Keltic Lodge, Ingonish Beach, Nova Scotia, Canada.

July 28, AIME, Adirondack Section, lead and talc mining and milling, Gouverneur, N. Y.

Aug. 19-24, Sixth International Symposium on Combustion, Yale University, New Haven, Conn.

Aug. 23-25, National Council of State Boards of Engineering Examiners, Hotel Statler, Los Angeles.

Aug. 25, AIME, Adirondack Local Section, golf, Tupper Lake, N. Y.

Sept. 4-11, 20th International Geological Congress, Mexico City.

Sept. 15, AIME, Colorado MBD Subsection, Glenwood Springs, Colo.

Sept. 16-21, ASTM, annual meeting and exhibit, Pacific area, Hotel Statler, Los Angeles.

Sept. 22, AIME, Adirondack Local Section, copper mining program, South Strafford, Vt.

Sept. 26-28, AIME Rocky Mountain Minerals Conference, Salt Lake City. Technical papers will represent all branches, including the Petroleum Div.

Oct. 1-4, American Mining Congress, Mining Show, Shrine Auditorium, Los Angeles.

Oct. 8-10, AIME, Institute of Metals Div., Carter Hotel, Cleveland.

Oct. 8-12, Union of Pan American Assns. of Engineers, fourth convention, Mexico City, Mexico.

Oct. 14-17, AIME, Petroleum Branch, Biltmore Hotel, Los Angeles.

Oct. 23-24, ASA, 38th annual meeting, Hotel Roosevelt, New York.

Oct. 25, 26, AIME-ASME Joint Solid Fuels Conference, *Future Role of Solid Fuels in an Expanding Economy,* Sheraton-Park Hotel, Washington, D. C.

Oct. 29-Nov. 1, Society of Exploration Geophysicists, 26th annual meeting, New Orleans.

Oct. 31-Nov. 2, Gulf Coast Assn. of Geological Societies, annual convention, Plaza Hotel, San Antonio, Texas.

Nov. 1-3, Geological Society of America, Minneapolis.

Nov. 1-3, New Mexico Mining Assn. and International Mining Days, Carlsbad, N. M.

Nov. 3, AIME, Adirondack Local Section, football game, Syracuse, N. Y.

Nov. 8-10, AIME, Northeastern Mining Branch Conference, Hershey, Pa. Lehigh Valley Section is host.

Nov. 12-15, Amer. Petroleum Inst., annual meeting, Minneapolis.

Dec. 5-7, AIME, Electric Furnace Steel Conference, Hotel Morrison, Chicago.

Feb. 7-9, Colorado Mining Assn., Denver.

Feb. 24-28, 1957, AIME Annual Meeting. Roosevelt and Jung hotels, New Orleans.

Mar. 10-16, EJC Second Nuclear Engineering and Science Congress, Convention Hall, Philadelphia.

Apr. 1-4, Amer. Assn. of Petroleum Geologists, annual meeting, Kiel Auditorium, St. Louis.

Apr. 8-10, AIME National Open Hearth Steel and Blast Furnace, Coke Oven, and Raw Materials Conferences, William Penn Hotel, Pittsburgh.

Apr. 11-13, AIME, Pacific Northwest Regional Conference, Portland, Ore.

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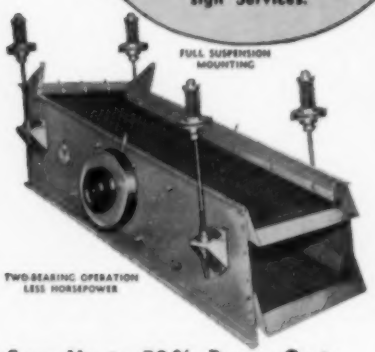

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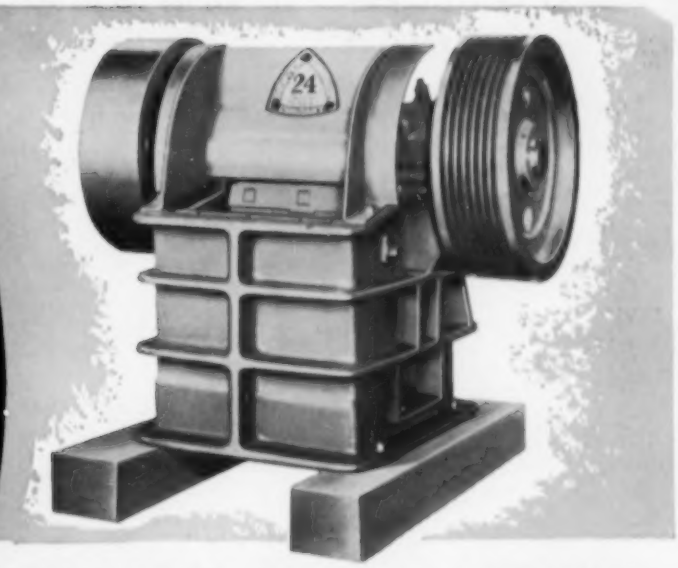

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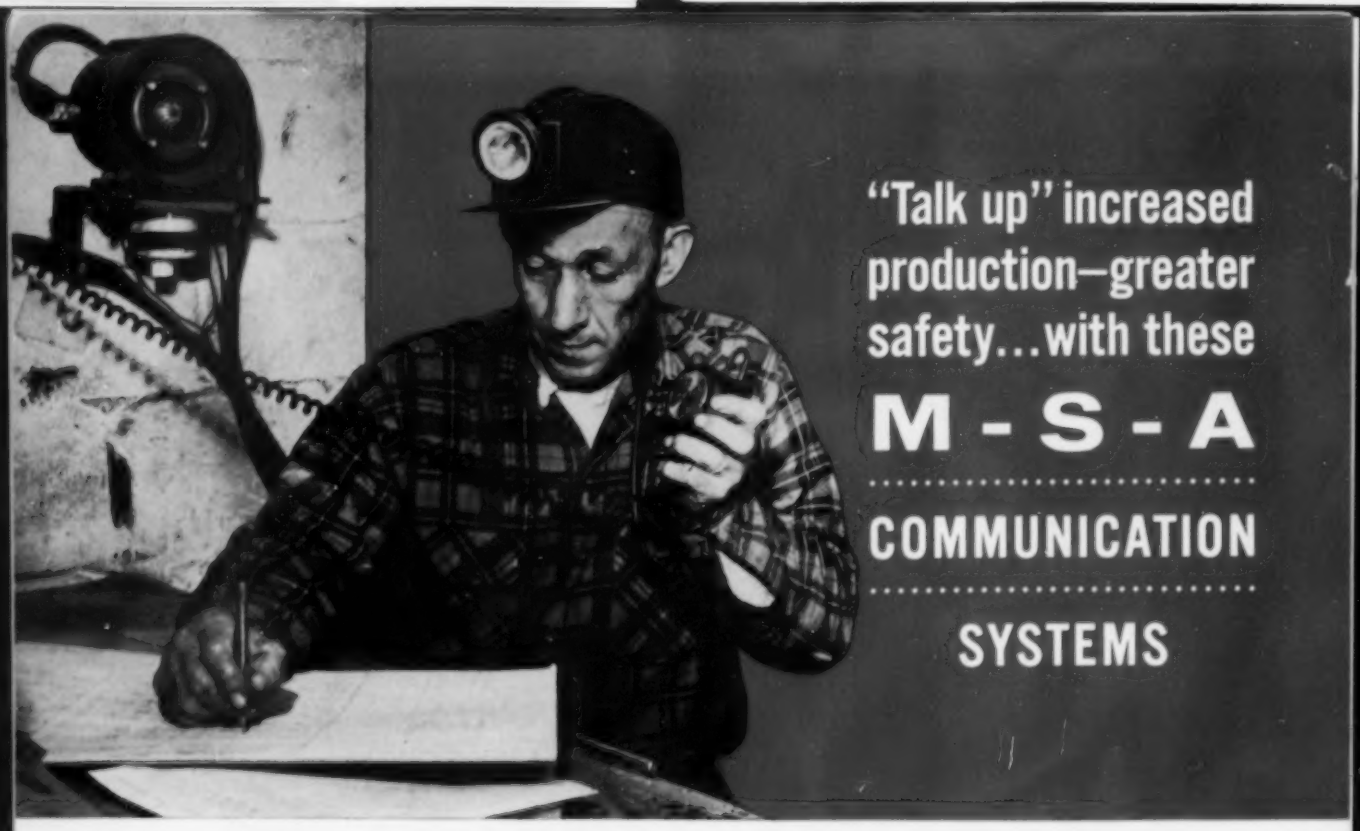

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